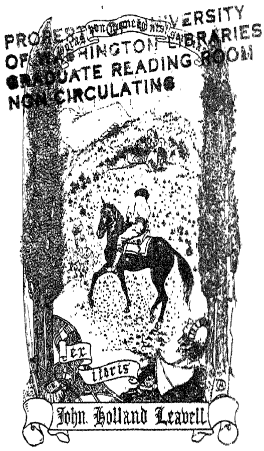

ex libris
John Holland Leavell

Dr H. Wedding.

See Biographical Notice, p. 538.

TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XL.

CONTAINING THE PAPERS AND DISCUSSIONS OF 1909.

NEW YORK, N. Y.:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.
1910.

PREFACE.

THIS volume contains all the proceedings, papers, and discussions of the Institute published during 1909, with the following exceptions:

1. Brief obituary notices of members and associates reported as deceased during the year 1909; Library accessions and requirements; notices of meetings of the Institute and of other societies; lists of proposed members and associates; changes of address of members; and other announcements of general but temporary interest, furnished to members in *Bulletin* Nos. 25 to 36, during the year 1909.

2. Account of the excursions and entertainments connected with the New Haven meeting, February, 1909,[1] and with the Spokane meeting, September, 1909.[2]

3. Year Book, containing a List of Members and Associates, revised to Jan. 1, 1910, 163 pages. Published in separate form and distributed with *Bulletin* No. 37, January, 1910.

4. The following papers, presented at the New Haven meeting:

A Rational Basis for the Conservation of Mineral Resources, by Joseph A. Holmes, Washington, D. C.[3]

The Mineral Wealth of America, by R. W. Raymond and W. R. Ingalls, New York, N. Y.[4]

The Iron-Ore Supply of the United States, by C. Willard Hayes, Washington, D. C.[5]

Biographical Notice of James Duncan Hague, by R. W. Raymond, New York, N. Y.[6]

The Concentration of Silver-Lead Ores at the Works of

[1] *Bulletin* No. 28, April, 1909, pp. 430 to 436.

[2] *Idem*, No. 36, December, 1909, pp. 1065 to 1118.

[3] *Idem*, No. 29, May, 1909, pp. 469 to 476.

[4] *Idem*, No. 27, March, 1909, pp. 249 to 264.

[5] *Idem*, No. 28, April, 1909, pp. 373 to 379.

[6] *Trans.*, xxxix., frontispiece and pp. 677 to 685.

Block 10 Co., Broken Hill, N. S. W., Australia, by V. F. Stanley Low, Broken Hill, N. S. W., Australia.[7]

A Sea-Level Canal at Panama—A Study of Its Desirability and Feasibility, by Henry G. Granger, Cartagena, Colombia, S. A.,[8] and the discussion thereof, by Lewis M. Haupt, A. Woodroffe Manton, John C. Oakes, R. R. Hancock, Gustav H. Schwab, H. L. Millner, W. L. Saunders, Charles Whiting Baker, John D. Evans, Ernest Howe, and Henry G. Granger.[9]

5. The following papers, presented at the Spokane meeting:

The Ventilating-System at the Comstock Mines, Nevada, by George J. Young, Reno, Nev.[10]

Review of Modern Cyanide Practice in the United States and Mexico, by S. F. Shaw, Los Angeles, Cal.[11]

6. The following papers, presented at a joint meeting of the American Society of Civil Engineers, the American Institute of Mining Engineers, the American Society of Mechanical Engineers, and the American Institute of Electrical Engineers:[12]

The Conservation of Water, by John R. Freeman.

The Conservation of Natural Resources by Legislation, by R. W. Raymond.

The Waste of Our Natural Resources by Fire, by Charles Whiting Baker.

Electricity and the Conservation of Energy, by Lewis B. Stillwell.

7. A few discussions referring to papers contained in Vol. XXXIX., which were received early in the year 1909, yet in time to be included in said volume.

On the other hand, this volume includes:

8. The following papers presented at the Chattanooga meeting, which were omitted from Vol. XXXIX. on account of lack of space:

Studies of Illinois Coals, by H. Foster Bain, Frank W. DeWolf, J. M. Lindgren, Perry Barker, George S. Rice, J. M. Snodgrass, A. Bement, W. F. Wheeler, and C. K. Francis.

---

7 *Bulletin* No. 33, September, 1909, pp. 763 to 793.

8 *Idem*, No. 25, January, 1909, pp. 1 to 37.

9 *Idem*, No. 31, July, 1909, pp. 623 to 666.

10 *Idem*, No. 35, November, 1909, pp. 955 to 1009.

11 *Idem*, No. 31, July, 1909, pp. 591 to 619.

12 *Idem*, No. 29, May, 1909, appendix.

The Clinton Iron-Ore Deposits in Alabama, by Ernest F. Burchard, Washington, D. C.

The Clinton Iron-Ore Deposits in Stone Valley, Huntingdon County, Pa., by J. J. Rutledge, Baltimore, Md.

The Clinton Iron-Ore Deposits in New York State, by D. H. Newland, Albany, N. Y.

Ozark Lead- and Zinc-Deposits: Their Genesis, Localization, and Migration, by Charles R. Keyes, Des Moines, Iowa.

Monazite and Monazite-Mining in the Carolinas, by Joseph Hyde Pratt, Chapel Hill, N. C., and Douglas B. Sterrett, Washington, D. C.

9. Discussions presented at the Pittsburg meeting, March, 1910, of papers contained in this volume which were received in time to be here printed and not held over for Vol. XLI. These discussions are:

Discussion of the paper of Audley H. Stow, Pressure-Fans *vs.* Exhaust-Fans, by R. V. Norris, Wilkes-Barre, Pa.

Discussion of the paper of Charles R. Keyes, Borax-Deposits of the United States, by A. M. Strong, Bishop, Cal.

Discussion of the paper of Edward W. Parker, The Conservation of Coal in the United States, by W. L. Saunders, New York, N. Y.

Discussion of the paper of James Douglas, Conservation of Natural Resources, by James Douglas, New York, N. Y.

Discussion of the paper of Albert F. J. Bordeaux, Cyaniding of Silver-Ores in Mexico, by Herbert A. Megraw, San Luis de la Paz, Guanajuato, Mexico.

Discussion of the paper of Messrs. Hofman and Hayward, Pan-Amalgamation: an Instructive Laboratory-Experiment, by George W. Riter, Salt Lake City, Utah; and reply by Messrs. Hofman and Hayward.

Discussion of the paper of Edmund D. North, Glass Mine-Models, by A. Scott Reid, London, England.

Reply to Mr. Buckley's discussion of the paper of Charles R. Keyes, Ozark Lead- and Zinc-Deposits: Their Genesis, Localization, and Migration, by Charles R. Keyes.

Reply to Messrs. Courtis and MacDonald's discussion of the paper of Charles R. Keyes, Genesis of the Lake Valley, New Mexico, Silver-Deposits, by Charles R. Keyes.

JOSEPH STRUTHERS,
Assistant Secretary and Editor.

CONTENTS.

PROCEEDINGS.

PAPERS.

DISCUSSIONS.

OFFICERS.

For the year ending February, 1910.

COUNCIL.*

PRESIDENT OF THE COUNCIL.

D. W. BRUNTON...DENVER, COLO.

(Term expires February, 1910.)

VICE-PRESIDENTS OF THE COUNCIL.

J. PARKE CHANNING..NEW YORK, N. Y.
FREDERICK W. DENTON.......................................PAINESDALE, MICH.
JOHN B. FARISH..NEW YORK, N. Y.

(Term expires February, 1910.)

W. C. RALSTON ....SAN FRANCISCO, CAL.
W. L. SAUNDERS..NEW YORK, N. Y.
H. V. WINCHELL ....MINNEAPOLIS, MINN.

(Term expires February, 1911.)

COUNCILORS.

B. F. FACKENTHAL, JR...EASTON, PA.
H. O. HOFMAN..BOSTON, MASS.
WALTER R. INGALLS...NEW YORK, N. Y.

(Term expires February, 1910.)

ARTHUR S. DWIGHT...NEW YORK, N. Y.
R. V. NORRIS...WILKES-BARRE, PA.
WILLIAM H. SHOCKLEY...TONOPAH, NEV.

(Term expires February, 1911.)

KARL E. EILERS..NEW YORK, N. Y.
ALEX. C. HUMPHREYS...NEW YORK, N. Y.
WILLET G. MILLER...TORONTO, CANADA.

(Term expires February, 1912.)

SECRETARY OF THE COUNCIL.

R. W. RAYMOND.................................29 W. 39th St., NEW YORK, N. Y.

(Term expires February, 1910.)

ASSISTANT SECRETARY AND EDITOR.

JOSEPH STRUTHERS..NEW YORK, N. Y.

CORPORATION.

JAMES GAYLEY, President; JAMES DOUGLAS, Vice-President;
R. W. RAYMOND, Secretary; FRANK LYMAN, Treasurer;
JOSEPH STRUTHERS, Assistant Secretary and Assistant Treasurer.

DIRECTORS.

JAMES GAYLEY, CHARLES KIRCHHOFF, FRANK LYMAN.

(Term expires February, 1910.)

JAMES DOUGLAS, JAMES F. KEMP, ALBERT R. LEDOUX.

(Term expires February, 1911.)

THEODORE DWIGHT, CHARLES H. SNOW, R. W. RAYMOND.

(Term expires February, 1912.)

Consulting Attorneys, Blair & Rudd, New York, N. Y.

---

* SECRETARY'S NOTE.—The Council is the professional body, having charge of the election of members, the holding of meetings (except business meetings), and the publication of papers, proceedings, etc. The Board of Directors is the body legally responsible for the business management of the Corporation, and is therefore, for convenience, composed of members residing in New York.

OFFICERS ELECTED AT ANNUAL MEETING, FEB. 15, 1910.

The list of officers on the opposite page is for the year 1909, the period covered by the contents of this volume of the *Transactions*. But the result of the election at the Annual Business Meeting, February, 1910, although strictly belonging to the next volume, is here published for the convenience of members.

The following officers were elected by vote of the members and associates in person or by proxy at the Annual Meeting, Feb. 15, 1910:

COUNCIL.

President of the Council.

D. W. Brunton, ... Denver, Col.

(To serve for one year. Term expires February, 1911.)

Vice-Presidents of the Council.

Benjamin B. Lawrence, ... New York, N. Y.
Joseph W. Richards, ... South Bethlehem, Pa.
Albert Sauveur, ... Cambridge, Mass.

(To serve for two years. Term expires February, 1912.)

Councilors.

Robert E. Jennings, ... New York, N. Y.
William Kelly, ... Vulcan, Mich.
Charles F. Rand, ... New York, N. Y.

(To serve for three years. Term expires February, 1913.)

Secretary of the Council.

R. W. Raymond, ... New York, N. Y.

(To serve for one year. Term expires February, 1911.)

Assistant Secretary and Editor (by appointment).

Joseph Struthers, ... New York, N. Y.

(To serve for one year. Term expires February, 1911.)

DIRECTORS OF THE CORPORATION.

James Gayley, Charles Kirchhoff, and Frank Lyman.

To serve for three years. Term expires February, 1913.)

The following are the officers of the Corporation for the year ending February, 1911:

President, James Gayley, New York, N. Y.
Vice-President, James Douglas, New York, N. Y.
Secretary, R. W. Raymond, New York, N. Y.
Treasurer, Frank Lyman, New York, N. Y.
Assistant Secretary and Assistant Treasurer, Joseph Struthers, New York, N. Y.

PAST OFFICERS.

Presidents.

David Thomas	1871
R. W. Raymond	1872–1874
A. L. Holley	1875
Abram S. Hewitt	1876
T. Sterry Hunt	1877
Eckley B. Coxe	1878–1879
William P. Shinn	1880
William Metcalf	1881
Richard P. Rothwell	1882
Robert W. Hunt	1883
James C. Bayles	1884–1885
Robert H. Richards	1886
Thomas Egleston	1887
William B. Potter	1888
Richard Pearce	1889
Abram S. Hewitt	1890
John Birkinbine	1891–1892
H. M. Howe	1893
John Fritz	1894
J. D. Weeks	1895
E. G. Spilsbury	1896
Thomas M. Drown	1897
Charles Kirchhoff	1898
James Douglas	1899–1900
E. E. Olcott	1901–1902
Albert R. Ledoux	1903–1904
James Gayley (Council)	1905
James Gayley (Corporation)	1905–1909
Robert W. Hunt (Council)	1906
John Hays Hammond (Council)	1907–1908
D. W. Brunton (Council)	1909

Secretaries.

Martin Coryell	1871–1872
Thomas M. Drown	1873–1884
R. W. Raymond	1884 ——

Treasurers.

J. Pryor Williamson	1871–1872
Theodore D. Rand	1872–1903
Frank Lyman	1903 ——

HONORARY MEMBERS.

PROF. RICHARD ÅKERMAN..Stockholm, Sweden.
PROF. RICHARD BECK .... Freiberg, Germany.
ANDREW CARNEGIE..New York, N. Y.
DR. JAMES DOUGLAS..New York, N. Y.
PROF. HATON DE LA GOUPILLIÈRE..Paris, France.
SIR ROBERT A. HADFIELD..London, England.
PROF. HANS HOEFER..Leoben, Austria.
PROF. HENRI LOUIS LE CHATELIER..Paris, France.
M. FLORIS OSMOND ....Saint Leu, France.
ALEXANDRE POURCEL..Paris, France.
DR. ING. H. C. EMIL SCHROEDTER..Düsseldorf, Germany.
JOHN E. STEAD..Middlesbrough, England.
JAMES M. SWANK (Associate) .... Philadelphia, Pa.
PROF. DIMITRY CONSTANTIN TSCHERNOFF..St. Petersburg, Russia.
CHARLES D. WALCOTT..Washington, D. C.

HONORARY MEMBERS (*Deceased*).

Year of Decease.

BELL, SIR LOWTHIAN..1904
CASTILLO, A. DEL..1895
CONTRERAS, MANUEL MARIA..1902
DAUBRÉE, A..1896
DROWN, THOMAS M..1904
GAETZSCHMANN, MORITZ..1895
GRUNER, L..1883
HUNT, T. STERRY..1892
KERL, BRUNO..1905
LE CONTE, JOSEPH..1901
LESLEY, J. P..1896
PATERA, ADOLPH..1890
PERCY, JOHN..1889
POSEPNY, FRANZ..1895
RICHTER, THEODOR..1898
ROBERTS-AUSTEN, W. C..1902
SERLO, ALBERT..1898
SIEMENS, C. WILLIAM..1883
THOMAS, DAVID..1882
TUNNER, PETER R. VON..1897
WEDDING, HERMANN 1908

LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO SEPTEMBER, 1909.

No.	Place.	Date.	Trans. Vol.	Trans. Page.
1.	Wilkes-Barre, Pa.*	May, '71	1	3
2.	Bethlehem, Pa	Aug., '71	1	10
3.	Troy, N. Y.	Nov., '71	1	13
4.	Philadelphia, Pa	Feb., '72	1	17
5.	New York, N. Y.*	May, '72	1	20
6.	Pittsburg, Pa	Oct., '72	1	25
7.	Boston, Mass	Feb., '73	1	28
8.	Philadelphia, Pa.*	May, '73	2	3
9.	Easton, Pa	Oct., '73	2	7
10.	New York, N. Y	Feb., '74	2	11
11.	St. Louis, Mo.*	May, '74	3	3
12.	Hazleton, Pa	Oct., '74	3	8
13.	New Haven, Conn	Feb., '75	3	15
14.	Dover, N. J.*	May, '75	4	3
15.	Cleveland, O	Oct., '75	4	9
16.	Washington, D. C	Feb., '76	4	18
17.	Philadelphia, Pa.†	June, '76	5	3
18.	Philadelphia, Pa	Oct., '76	5	19
19.	New York, N. Y	Feb., '77	5	27
20.	Wilkes-Barre, Pa.*	May, '77	6	3
21.	Amenia, N. Y	Oct., '77	6	10
22.	Philadelphia, Pa	Feb., '78	6	18
23.	Chattanooga, Tenn.*	May, '78	7	3
24.	Lake George, N. Y	Oct., '78	7	103
25.	Baltimore, Md.*	Feb., '79	7	217
26.	Pittsburg, Pa	May, '79	8	3
27.	Montreal, Canada	Sept., '79	8	121
28.	New York, N. Y.*	Feb., '80	8	275
29.	Lake Superior, Mich	Aug., '80	9	1
30.	Philadelphia, Pa.*	Feb., '81	9	275
31.	Staunton, Va	May, '81	10	1
32.	Harrisburg, Pa	Oct., '81	10	119
33.	Washington, D. C.*	Feb., '82	10	225
34.	Denver, Colo	Aug., '82	11	1
35.	Boston, Mass.*	Feb., '83	11	217
36.	Roanoke, Va	June, '83	12	3
37.	Troy, N. Y	Oct., '83	12	175
38.	Cincinnati, O.*	Feb., '84	12	447
39.	Chicago, Ill	May, '84	13	1
40.	Philadelphia, Pa	Sept., '84	13	285
41.	New York, N. Y.*	Feb., '85	13	585
42.	Chattanooga, Tenn	May, '85	14	1
43.	Halifax, N. S	Sept., '85	14	307
44.	Pittsburg, Pa.*	Feb., '86	14	587
45.	Bethlehem, Pa	May, '86	15	lxiii.
46.	St. Louis, Mo	Oct., '86	15	lxx.
47.	Scranton, Pa.*	Feb., '87	15	lxxvii.
48.	Utah and Montana	July, '87	16	xvii.
49.	Duluth, Minn	July, '87	16	xxiv.
50.	Boston, Mass.*	Feb., '88	16	xxviii.
51.	Birmingham, Ala	May, '88	17	xix.
52.	Buffalo, N. Y	Oct., '88	17	xxiv.
53.	New York, N. Y.*	Feb., '89	17	xxxi.
54.	Colorado	June, '89	18	xvii.
55.	Ottawa, Canada	Oct., '89	18	xxiv.
56.	Washington, D. C.*	Feb., '90	18	xxx.
57.	New York, N. Y	Sept., '90	19	vii.
58.	New York, N. Y.*	Feb., '91	19	xxv.
59.	Cleveland, O	June, '91	20	xvi.
60.	Glen Summit, Pa	Oct., '91	20	lxi.
61.	Baltimore, Md.*	Feb., '92	21	xix.
62.	Plattsburgh, N. Y	June, '92	21	xxxiii.
63.	Reading, Pa	Oct., '92	21	xliv.
64.	Montreal, Canada*	Feb., '93	21	lii.
65.	Chicago, Ill	Aug., '93	22	xiii.
66.	Virginia Beach, Va.*	Feb., '94	24	xvii.
67.	Bridgeport, Conn	Oct., '94	24	xxxv.
68.	Florida‡	Mar., '95	25	xix.
69.	Atlanta, Ga	Oct., '95	25	xxxiii.
70.	Pittsburg, Pa.*	Feb., '96	26	xvii.
71.	Colorado	Sept., '96	26	xxix.
72.	Chicago, Ill	Feb., '97	27	xvii.
73.	Lake Superior	July, '97	27	xxx.
74.	Atlantic City, N. J.*	Feb., '98	28	xvii.
75.	Buffalo, N. Y	Oct., '98	28	xxxvi.
76.	New York, N. Y.*	Feb., '99	29	xvii.
77.	California	Sept., '99	29	xlix.
78.	Washington, D. C.*	Feb., '00	30	xix.
79.	Canada	Aug., '00	30	xlv.
80.	Richmond, Va.*	Feb., '01	31	xix.
81.	Mexico	Nov., '01	32	cxviii.
82.	Philadelphia, Pa. §	May, '02	33	xxxv.
83.	New Haven, Conn	Oct., '02	33	xlvii.
84.	Albany, N. Y.*	Feb., '03	34	xxiii.
85.	New York, N. Y	Oct., '03	34	lxi.
86.	Atlantic City, N. J.*	Feb., '04	35	xxiii.
87.	Lake Superior	Sept., '04	35	xlii.
88.	Washington, D. C	May, '05	36	xlii.
89.	British Columbia	July, '05	36	liii.
90.	Bethlehem, Pa	Feb., '06	37	xli.
91.	London, England	July, '06	37	xlviii.
92.	New York, N. Y	April, '07	38	lii.
93.	Toronto, Canada	July, '07	38	lix.
94.	New York, N. Y	Feb., '08	39	xli.
95.	Chattanooga, Tenn	Oct., '08	39	xlviii.
96.	New Haven, Conn	Feb., '09	40	xli.
97.	Spokane, Wash	Sept., '09	40	xlviii.

* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

‡ Begun in February at New York City, for the election of officers, and adjourned to Florida.

§ " " " " " " " " " " " " " to Philadelphia.

PUBLICATIONS.

THE publications of the Institute comprise:

TRANSACTIONS.

The volumes of *Transactions*, which are published annually, contain the list of officers, rules, etc., the Proceedings, and the papers revised for final publication. (In this revision, after the preliminary publication, authors are permitted to use the largest liberty; and the changes and additions made in papers are sometimes important. It should be borne in mind by those who study or quote a paper in the preliminary edition, that they may not have in that form the ultimate and deliberate expression of the author's views. It should be added, however, that in the majority of cases there are no important changes.) These volumes are for sale as follows, in paper covers:

Vols. I. to IV. (inclusive), each, ...	$3.00
Vols. V. to VIII. (inclusive), each, ...	4.00
Vol. IX., ...	10.00
Vol. X. (a small supply on hand, which will be sold only with complete sets, at a price of $10.00), ...	
Vols. XI. to XXIX. (inclusive), each, ...	5.00
Vols. XXX. and XXXI., each, ...	6.00
Vol. XXXII., ...	5.00
Vols. XXXIII. to XL. (inclusive), each, ...	6.00
Half-morocco binding, $1 extra per volume.	
Complete set of *Transactions*, Vols. I. to XL., inclusive, half-morocco binding (freight prepaid), ...	248.00

BULLETIN.

Per annum, ...	10.00
(To members of the Institute, public libraries, educational institutions and technical societies, $5.00.)	
Single numbers, ...	1.00
(To members of the Institute, public libraries, etc., $0.50.)	

Index, Vols. I. to XXXV. (inclusive).—This volume, an octavo of 706 pages, affords a ready and complete reference to any subject treated or alluded to in the *Transactions*, Vols. I. to XXXV., inclusive. The names of persons, mines, works, towns, etc., have been included; and abundant cross-references and classified sub-headings have been added to facilitate rapid consultation.

The Institute maintains at more than a hundred important mining centers throughout the world, free sets of its *Transactions*, open for consultation without fee, to all suitable applicants. Hence, the value of this index is by no means limited to individual possessors of complete sets of the *Transactions*. Moreover, the title of a paper, or the record of any remarks concerning a subject, being found in the Index, the Secretary's office of the Institute will supply upon written application any desired information as to the nature and length of said paper, whether it can be supplied in separate pamphlet form, etc.

Bound in cloth, $5.00, half-morocco, ... $6.00

SPECIAL EDITIONS.

"*The Genesis of Ore-Deposits*," comprising the famous treatise of the late Professor Franz Pošepný, with the successive discussions thereof by Le Conte, Blake, Winchell, Church, Emmons, Becker, Cazin, Rickard and Raymond (all of which were published in Volumes XXIII. and XXIV. of the *Transactions* of the Institute, and subsequently in the special "Pošepný Volume," issued by the Institute); also, later papers by Van Hise, Emmons, Weed, Lindgren, Vogt, Kemp, Blake, Rickard and others, and the discussions of these papers by De Launay, Beck, and many others (some of these were included in Volume XXX. and the remainder appeared in Volume XXXI.); also a complete bibliography of Institute papers and discussions on this subject from 1871 to the year 1902.

The original Pošepný volume comprised 265 pages, and was sold for $2.50, at which price the edition was long since exhausted. The present volume is an octavo of 825 pages.

Bound in cloth, $6.00, half-morocco, ... $7.00

"*The Evolution of Mine-Surveying Instruments.*" This is a volume of about 400 pages, containing the original paper of Mr. Dunbar D. Scott on that subject (*Transactions*, XXVIII.), first published in 1898, together with later papers, continuing the same subject, and discussions thereof, by Hoskold, Lyman, Davis and many others.

Bound in cloth, $3.50, half-morocco, ... $4.50

Year Book, containing List of Members, Rules, etc., paper, to Members of the Institute, $0.50; to others, . . . 1.00

Glossary of Mining and Metallurgical Terms (1881), cloth, . $1.00
Spanish-American Mining and Metallurgical Glossary, bound in leather, pocket-size, 96 pages, ... 0.75
Chart for the Solution of Kutter's Formula, on cloth, . 0.50
Papers on the Conservation of Natural Resources, presented at the Joint Meeting of the four National Engineering Societies, March 24, 1909, by J. R. Freeman, R. W. Raymond, C. W. Baker, and L. B. Stillwell, . 0.25

Pamphlets.

1. The Minutes of the Proceedings of each Meeting.

2. Such of the papers presented or read by title at each Meeting as are furnished by the authors and approved by the Council for full publication. (In nearly all cases in which papers, the titles of which appear in the Proceedings, are not subsequently published, they have been withdrawn by the authors.) These papers are published separately in pamphlet form, and are marked "subject to revision." Beyond the edition distributed in the *Bulletin*, without charge, to members and associates, a small supply is retained to meet subsequent demand. There are no copies on hand of papers read before 1880. The stock is nearly complete from 1880. These papers are for sale at the office of the Secretary, or are sent to purchasers, charges paid, on receipt of the price, as follows:

No. of Pages.	Single Copies.	10 Copies.	20 Copies.
24 or less	$0.25	$2.00	$3.50
25 to 48	0.30	2.50	4.50
49 to 80	0.40	3.25	5.25
81 to 96	0.45	3.50	6.00
97 to 128	0.50	3.75	6.25
129 to 144	0.55	4.00	6.50
145 to 160	0.60	4.25	6.75
161 to 176	0.65	4.50	7.00

Papers with folders and inserted plates subject to special price.

Authors' Edition of Pamphlets.

Extra copies of pamphlets, if ordered before the printing of the *Bulletin*, will be furnished to members of the Institute at special rates, which will be given on application to the Assistant Secretary, Joseph Struthers, 29 West 39th St., New York, N. Y.

CONSTITUTION.

[ADOPTED FEB. 21, 1905.]

ARTICLE I.

NAME AND OBJECT.

SEC. 1. This Institute is incorporated under the Membership Corporation Law of the State of New York; its corporate name is AMERICAN INSTITUTE OF MINING ENGINEERS; and its objects are such as are stated in its Certificate of Incorporation.

ARTICLE II.

MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: (1) Members; (2) Honorary Members; (3) Associates; and (4) Honorary Associates. Only Members and Associates residing within the United States of America, Republic of Mexico and Dominion of Canada shall be entitled to vote at the meetings of the Institute.

SEC. 2. All Members, Honorary Members, Associates and Honorary Associates of the American Institute of Mining Engineers as the same existed on the day of the incorporation of this Institute, are Members, Honorary Members, Associates and Honorary Associates, respectively, of this Corporation.

SEC. 3. The following classes of persons shall be eligible for membership in the Institute, namely: as Members and Honorary Members, all professional mining engineers, geologists, metallurgists or chemists, and all persons practically engaged in mining, metallurgy or metallurgical engineering; as Associates and Honorary Associates, all persons desirous of being connected with the Institute who, in the opinion of the Council, are suitable.

SEC. 4. Every candidate for election as a Member or Associate of the Institute must be proposed for election by at least three Members or Associates; must be approved by the Committee on Membership, as prescribed in the By-Laws; and must be elected by the Council. Not less than three-fourths of the votes cast shall be necessary to an election. Every person so elected shall become a Member or Associate, as the case may be, upon payment of his first dues as hereinafter prescribed. Each candidate for Honorary Member or Honorary Associate, must be recommended by at least ten Members or Associates; must be approved by the Council; and must be elected by ballot at a meeting of the Board of Directors by the unanimous vote of all the Directors present; provided, however, that the number of Honorary Members and Honorary Associates shall not at any time exceed twenty.

SEC. 5. If any person elected a Member or Associate does not, within sixty days after notice of his election, accept the same and pay his initiation fee and dues for the current year, his election may be cancelled at the discretion of the Council.

SEC. 6. The Council may at any time change the classification of a person elected as an Associate so as to make him a Member, or vice versa. All Members and Associates shall be equally entitled to the privileges of membership, provided that Honorary Members, Honorary Associates, and Members and Associates whose Post-Office addresses shall be outside of the United States, Mexico and Canada, shall not be entitled to vote.

ARTICLE III.

DUES.

SEC. 1. The dues of Members and Associates shall be Ten Dollars per annum, payable in advance on the first day of each Calendar year. Each newly elected Member or Associate shall pay, when notified of election, an initiation fee of Ten Dollars in addition to the dues for the current year. Honorary Members and Honorary Associates shall not be liable to initiation fee or dues. Any Member or Associate in arrears for one year may, at the discretion of the Council, be deprived of the receipt of publications or stricken from the list of Members, provided that he may be restored to membership by the Council on payment of all arrears or may be again proposed and elected after an interval of three years.

SEC. 2. Any Member or Associate not in arrears may become, by the payment of One Hundred and Fifty Dollars at one time, a Life Member or Associate; and shall not be liable thereafter to annual dues.

ARTICLE IV.

BUSINESS MEETINGS OF THE INSTITUTE.

SEC. 1. The annual meeting of the Institute for the election of Directors and transaction of other business shall take place on the third Tuesday in February in each year. A report of the financial condition of the Institute and an abstract of the accounts shall be furnished by the Directors, and presented at each annual meeting.

SEC. 2. Special business meetings of the Institute may be held at such times and places as the Board of Directors may appoint, upon notice to all Members and Associates entitled to vote, directed to each at his last known Post-Office address, and mailed in the City of New York not less than twenty days before the date fixed for such meeting.

SEC. 3. At all business meetings of the Institute the presence of nine Members and Associates shall constitute a quorum.

SEC. 4. At all business meetings of the Institute Members and Associates may vote either in person or by proxy, but no Member or Associate in arrears since the last annual meeting shall be entitled to vote.

ARTICLE V.

OTHER MEETINGS OF THE INSTITUTE.

SEC. 1. All meetings of the Institute other than business meetings shall be held at such times and places as the Council may appoint. Notice of all such meetings shall be given to all Members and Associates by mail.

ARTICLE VI.

Directors and Officers.

Sec. 1. The business and financial affairs of the Institute shall be managed by a Board of Directors, who shall be elected at the annual meeting in the manner prescribed in the Certificate of Incorporation.

Sec. 2. The officers of the corporation shall be a President, Vice-President, Secretary and Treasurer, who shall be elected by the Directors from among their number. All such officers shall be elected at the first meeting of the Board of Directors after each annual meeting of the corporation, and shall hold office for one year or until their successors are elected and qualify.

The duties of all officers shall be such as usually pertain to their offices, respectively, together with such other duties as may from time to time be prescribed for them by the By-Laws. The Treasurer shall give a bond for the faithful performance of his duties in a sum to be fixed by the Board of Directors, but at the expense of the Institute.

Sec. 3. In the event of a vacancy occurring in the Board of Directors by death, resignation or otherwise, the remaining members of the Board may, by a majority vote, elect a successor to fill the vacancy, who shall continue in office until the next annual meeting or until his successor shall have been chosen.

Sec. 4. The Board of Directors may, in its discretion, declare the place of any Director vacant, on his failure for any reason, to attend three successive meetings of the Board. Any Director who shall under this section or in any other manner cease to be a member of the Board shall, at the same time, be held to have vacated any other office to which he shall previously have been elected; and the Board shall elect a new incumbent to the said vacant office.

Sec. 5. The Board of Directors may from time to time appoint from their own number standing and special committees, and may delegate to such committees such duties as they may see fit.

ARTICLE VII.

Meetings of the Board of Directors.

Sec. 1. A regular meeting of the Board of Directors for the election of officers and the transaction of other business shall be held on the third Tuesday in February in each year, after the adjournment of the annual meeting of the Institute.

Sec. 2. Special meetings of the Board of Directors, at which any business may be transacted, may be called to meet at any time at the office of the Institute in the City of New York, by notice in writing mailed at least five days before the meeting, by the Secretary to each member of the Board at his last known Post-Office address, signed either by the President or the Vice-President or by three members of the Board.

Sec. 3. At all meetings of the Board of Directors the presence of five members shall constitute a quorum.

ARTICLE VIII.

The Council.

Sec. 1. The professional, technical, scientific and social interests of the Institute shall be committed to the supervision of a Council composed of a President

of the Council, six Vice-Presidents of the Council, a Secretary of the Council and nine Councilors, who shall be elected from among the Members and Associates of the Institute in the manner hereinafter prescribed. Members of the Council may or may not be members of the Board of Directors.

SEC. 2. The President of the Council shall be elected for one year, and no person shall be eligible for immediate re-election to this office who shall have held the same for two consecutive years.

After the first year Vice-Presidents of the Council shall be elected to serve for two years, and Councilors shall be elected to serve for three years. No Vice-President of the Council or Councilor shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. The Secretary of the Council shall be elected annually.

SEC. 3. At the first annual meeting, to be held in the year 1905, there shall be elected a President of the Council to serve for one year, a Secretary of the Council to serve for one year, three Vice-Presidents of the Council to serve for one year, three Vice-Presidents of the Council to serve for two years, three Councilors to serve for one year, three Councilors to serve for two years, and three Councilors to serve for three years. At each subsequent annual meeting there shall be elected a President of the Council to serve for one year; a Secretary of the Council to serve for one year; three Vice-Presidents of the Council to serve for two years; and three Councilors to serve for three years. The term of office of all Members of the Council shall continue until the adjournment of the meeting at which their successors are elected.

SEC. 4. Vacancies in the Council may occur by death or resignation; or the Council may, by the vote of a majority of all its members, declare the place of any officer or member of the Council vacant, on his failure for one year, from inability or otherwise, to attend the regular meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *provided* that the said appointment shall not render such person ineligible for election to the Council at the next meeting.

SEC. 5. The presence of five members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or any business coming within the authority of the Council may be transacted at a regularly-called meeting thereof, at which less than a quorum may be present, subject to the approval of a majority of the Council subsequently given in writing to the Secretary and recorded by him with the minutes.

SEC. 6. The election of the Council shall take place at the regular annual meeting of the Institute. Nominations for members of the Council may be sent in writing to the Secretary accompanied with the names of the proposers at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before said meeting, mail to every Member or Associate entitled to vote a list of all nominations for each office so received, together with the names of the persons ineligible for election to each office; and if the Council or a Committee thereof, appointed for the purpose, shall have recommended any nomination, such recommendation may also be sent to the Members and Associates with the list of all nominations made.

ARTICLE IX.

Meetings of the Council.

Sec. 1. Meetings of the Council shall be held at such times and places as the President of the Council or one of the Vice-Presidents of the Council may appoint.

Sec. 2. A meeting of the Council may be held on the day of the annual meeting of the Institute without previous notice. Written notice of all other meetings of the Council, specifying the time and place of such meeting, signed by the Secretary, shall be mailed to every member of the Council at his last known Post-Office address at least ten days before the date of the meeting.

ARTICLE X.

Papers and Publications.

Sec. 1. The Council shall have power to decide as to the acceptance and publication of any professional papers presented to the Institute, subject to such conditions as the Board of Directors may prescribe.

Sec. 2. The copyright of all professional papers communicated to and accepted by the Institute shall be vested in it, unless otherwise expressly agreed between the Council and the author. The Institute shall not assume responsibility for any statements of fact or opinion advanced in the papers or discussions at its meetings. Neither the Council nor the Institute shall officially approve or disapprove any technical or scientific opinion or any proposed enterprise, outside of the management of the meetings, discussions and publications of the Institute, and the conduct of its business affairs by the Board of Directors.

Sec. 3. Special Committees may from time to time be appointed by the Council to make investigations and prepare reports for presentation to the Institute, but no action shall be taken binding the Institute for or against the conclusions embodied in any such reports.

ARTICLE XI.

Suspensions and Expulsions.

Sec. 1. Any member of the Institute who shall be convicted of a crime involving, in the opinion of the Board of Directors, moral turpitude, shall, upon the passage by the Board of Directors of a resolution declaring the crime for which he has been convicted to be of such character, be thereupon dropped from membership in this Institute.

Sec. 2. Any member of the Institute may be suspended or expelled for misconduct by the Board of Directors, after charges setting forth such misconduct shall have been prepared by the Council and filed in writing with the Board. Upon the receipt of such charges in writing, the Board may, in its discretion, suspend such member pending a hearing and determination thereupon. As soon as may be after the receipt of such charges, the Board shall fix a date for a hearing thereupon and shall give to the accused member notice thereof in writing, mailed to him at his last known Post-Office address not less than thirty days before said date, accompanied by a full copy of the charges and a copy of the second, third and fourth sections of this article.

Sec. 3. Upon the day fixed for the hearing, the accused member may appear before the Board, either in person or by an accredited representative; hear any

witnesses who may be called in support of the charges and at his option cross-examine the same; and hear read any documentary evidence offered in support of the charges. The accused may, in his discretion, produce and examine witnesses in his defence, and submit documentary evidence, including a statement from himself in writing. After the conclusion of the hearing, the Board of Directors shall consider and vote to approve or disapprove the charges. If the Board shall, by a vote of two-thirds of its members, declare the charges sustained, it may suspend the member for a stated period or expel him.

SEC. 4. If the accused member shall not appear at the hearing, and shall within three months thereafter file with the Board an affidavit stating that he had not received notice of the charges against him in time to enable him to present his defence, the Board shall fix a date for a re-hearing within three months from the receipt of such affidavit and shall immediately notify the accused member by mail of such date. Upon the re-hearing, the accused shall have the same privilege of presenting his defence as he would have had upon the original hearing; and after the defence is presented, the Board shall take a new vote upon the charges, the result of which shall be conclusive.

SEC. 5. All interests in the property of the Institute of persons resigning, or otherwise ceasing to be Members or Associates, shall vest in the Institute.

ARTICLE XII.

AMENDMENTS.

SEC. 1. This Constitution or any Article or Section thereof may be amended at any annual meeting by a two-thirds vote of all the members present in person or by proxy, *provided* that notice of the proposed amendment shall have been given in writing at a previous meeting, and *provided also* that the amendment or amendments so adopted shall have been printed and mailed to all Members and Associates not later than thirty days before the annual meeting. Any amendment or amendments approved by a majority of the votes cast shall be deemed to have been adopted, and shall become a part of this Constitution. The Secretary shall forthwith print and distribute to Members and Associates an announcement of the result of said vote, and if any amendment or amendments shall have been adopted, a copy of the section or sections so amended.

BY-LAWS.

[ADOPTED FEB. 21, 1905. AMENDED FEB. 20, 1906, NOV. 16, 1906, AND JAN. 5, 1909.]

I. PRESIDING OFFICERS.

At all Business meetings of the Institute the President, or, in his absence, the Vice-President, or, in the absence of both of them, any other member of the Board of Directors to be chosen by the meeting, shall preside.

At all other meetings of the Institute the President of the Council or, in his absence, one of the Vice-Presidents, if present, shall preside.

II. ORDER OF BUSINESS.

At each Business meeting of the Institute the order of business shall be as follows:

1. Reading of minutes of preceding meeting.
2. Report of the President.
3. Report of the Treasurer.
4. Report of the Secretary.
5. Election of Directors.
6. Election of Members of the Council.
7. Reports of Standing Committees.
8. Reports of Special Committees.
9. Special Orders.
10. Miscellaneous business.

This order of business may be changed by a vote of a majority of the Members and Associates present in person or by proxy.

The usual parliamentary rules shall govern all meetings of the Institute except in cases otherwise provided by the Constitution or the By-Laws.

At all sessions of the Institute other than business meetings, the order of proceedings and the time of adjournment shall rest in the discretion of the presiding officer.

III. SECRETARY.

The Secretary shall keep a record of the proceedings of all meetings of the Institute. He shall be custodian of the Corporate Seal, of the Minute Books, and of all Legal Documents belonging to the Institute. He shall conduct, on behalf of the Institute, all correspondence relating to business matters, except such as pertains directly to the office of the Treasurer.

He shall notify all officers and Directors and Members of the Council, and all Members of Committees of their election and appointment; shall issue notices of all meetings of the Board, and of the annual and other meetings of the Institute; and shall, in calling special meetings of the Directors, specify the object of such meeting.

IV. SECRETARY OF THE COUNCIL.

The Secretary of the Council shall act as the Clerk of that body at all of its meetings and at all meetings of the Institute called for the discussion of professional, technical or scientific matters, or for any other purpose than the transaction of business.

He shall be custodian of all technical or scientific papers submitted to the In-

stitute for its consideration, shall have charge of the editing and printing of all material published by the Institute, and of the distribution thereof. On the first day of May following the year in which each volume of *Transactions* is printed, he shall turn over to the Library Committee all copies of the same not theretofore distributed by him. He shall have charge of all the correspondence of the Institute relating to other than business affairs.

The Secretary of the Council shall receive a salary to be fixed by the Board of Directors. He may appoint an Assistant with the title of Editor, who shall likewise receive a salary to be fixed by the Board of Directors.

The Secretary of the Council may or may not be the same person as the Secretary of the Institute.

V. Assistant Secretary.

The Secretary may, with the approval of the Board of Directors, appoint an Assistant to whom both he and the Secretary of the Council may delegate such of his or their duties as he or they may see fit. This Assistant Secretary shall receive such salary as shall be fixed by the Board of Directors, which shall cover his services both to the Secretary and to the Secretary of the Council.

VI. Treasurer.

The Treasurer shall collect and, under the direction of the Board of Directors, shall disburse all funds of the Institute. He shall keep regular accounts in books belonging to the Institute, which shall be open to any member of the Board of Directors. He shall report in writing at each annual meeting of the Institute and at every meeting of the Board of Directors at which such report shall be called for, the balance of money on hand, and any existing appropriation which may affect the same.

His accounts shall be audited annually by a Committee of three Members or Associates to be appointed by the President at least thirty days prior to the annual meeting in each year, which Committee shall report thereon at such annual meeting.

The Treasurer may, at his discretion, place funds of the Institute, not at any time exceeding $5,000, in a special account in a Bank or Trust Company, subject to the draft of the Assistant Treasurer, and may delegate to the Assistant Treasurer the duty of paying, out of this account, the current expenses of the Institute.

The Treasurer shall be solely responsible to the Institute for all moneys received, whether the same are entrusted to the Assistant Treasurer or not.

VII. Assistant Treasurer.

The Treasurer may appoint, with the approval of the Board of Directors, an Assistant Treasurer, to whom he may delegate the duty of conducting the correspondence incidental to the office of Treasurer, of receiving and depositing in bank to the credit of the Institute all moneys received, and of paying, out of the special account upon which he may be authorized to draw, the necessary expenses of the Institute. The Treasurer may require of him a bond, running to the Treasurer personally, in an amount not exceeding $5,000, the expense of which shall be borne by the Institute.

The Assistant Treasurer shall receive such compensation as shall be fixed by the Board of Directors.

The offices of the Assistant Secretary and of the Assistant Treasurer may, if

so desired by both the Secretary and the Treasurer and approved by the Board of Directors, be united in the same person, who shall then receive the salary of both offices.

The Assistant Treasurer may, with the approval of the Board of Directors, employ such persons as are necessary to constitute a clerical and office force for himself, the Assistant Secretary and the Secretary of the Council, at such salaries as shall be approved by the Board of Directors. He shall, if the offices of Assistant Secretary and Assistant Treasurer be united in the same person, be the immediate superior of all such employees, unless the Secretary of the Council or the Treasurer be present, in which event either of them shall be the superior of all employees, including their respective assistants.

VIII. Standing Committees.

The Standing Committees of the Institute shall be three in number, known respectively as the Finance Committee, the Library Committee and the Committee on Membership.

The Finance Committee and the Library Committee shall each consist of three members of the Board of Directors, and shall be appointed by the President at the first meeting of the Board, after the annual meeting in each year.

The Committee on Membership shall consist of five Members of the Council, and shall be appointed by the President of the Council, at the first meeting of the Council after the first annual meeting in each year.

IX. Finance Committee.

It shall be the duty of the Finance Committee to inquire into and examine the financial condition of the Institute, and to consider ways and means of increasing its revenues and of limiting its expenses. It shall report from time to time to the Board as often as it may deem expedient, and whenever it shall be directed so to do; and the Treasurer shall at all times furnish it with such statements and information as it may desire.

It shall determine the investment of such surplus moneys as shall from time to time accrue to the Institute. It shall, at least once in each year, examine the securities belonging to the Institute in the custody of the Treasurer, and report thereon to the Board.

It may, at any time, examine the books and vouchers of the Treasurer and Assistant Treasurer.

The Treasurer shall not be a member of the Finance Committee, but shall attend the meetings of the same if requested to do so.

X. Library Committee.

The Library Committee shall be the custodian of all books in the Institute Library and of additions thereto; also of all back numbers of the *Transactions* of the Institute. It shall, on the first day of May, of each year, receive from the Secretary of the Council, and receipt for same to him, all the volumes of *Transactions* for the preceding year, not then distributed by said Secretary.

It shall cause to be kept, under the direction of the Assistant Secretary, a catalogue of all books in the Library and an account in ledger form of all volumes of *Transactions* in its custody, in which shall be charged to it all volumes delivered to it, and in which shall be credited all volumes taken from its custody for sale or for any other purpose.

The receipts from the sale of any volume of *Transactions* taken from the custody of the LIBRARY COMMITTEE shall be credited to the LIBRARY COMMITTEE on the books of the Treasurer, and devoted to the general purposes of the Institute.

XI. Committee on Membership.

All nominations for Members or Associates of the Institute shall be submitted to and passed upon by the COMMITTEE ON MEMBERSHIP, who shall report thereon to the Council. It shall receive and consider all communications respecting candidates, and shall make diligent inquiry as to the character and qualifications of each one. Its proceedings shall be secret and confidential.

No member of the Committee shall propose any candidate.

XII. Election of Members.

After the COMMITTEE ON MEMBERSHIP shall have reported to the Council its conclusions as to the acceptability of each candidate, the Council shall vote upon the same.

Two negative votes of members of the Council present shall prevent the election of any candidate. No person shall be proposed for election to the Institute within one year after his name shall have been rejected by the Council.

XIII. United Engineering Society.

The Board of Directors shall, at its first meeting after the adoption of these By-Laws, designate three Members or Associates of this Institute to be representatives of this Institute upon the Board of Trustees of the UNITED ENGINEERING SOCIETY, making at the same time provision for the expiration of the terms of office of said representatives, as provided in the By-Laws of the said UNITED ENGINEERING SOCIETY.

At the last meeting of the Board of Directors prior to the first day of each January thereafter, the Board shall designate a Member or Associate of this Institute to be a representative of this Institute upon the Board of Trustees of the said UNITED ENGINEERING SOCIETY for a period of three years beginning at the next ensuing annual meeting of said Society.

At any time when a vacancy shall occur in the representation of this Institute in the Board of Trustees of said Society, by reason of the death, resignation or removal of any such representative therein, the Board of Directors of this Institute shall designate a Member or Associate to fill such unexpired term.

XIV. Publications.

The publications of the Institute shall include a periodical, called the *Bulletin* of the American Institute of Mining Engineers, which shall contain reports of proceedings, professional papers, notices, and other matter of interest to members. From the annual dues paid by each Member or Associate, five dollars shall be deducted and applied as a subscription to the *Bulletin* for the year covered by such payment.

XV. Amendments.

These By-Laws may at any time be altered or amended by a vote of two-thirds of the Board of Directors, or by the Members, at a business meeting of the Institute, in the same manner provided for amendments of the Constitution in Article XII. thereof.

ANNUAL MEETING OF THE INSTITUTE.

At the Annual Business Meeting of the Institute, held Feb. 16, 1909, the following person were elected:

Council.

President.

(To serve for one year.)

D. W. Brunton, ... Denver, Colo.

Vice-Presidents.

(To serve for two years.)

W. L. Saunders, ... New York, N. Y.
H. V. Winchell, ... Minneapolis, Minn.
W. C. Ralston, ... San Francisco, Cal.

Secretary.

(To serve for one year.)

R. W. Raymond, ... New York, N. Y.

Councilors.

(To serve for three years.)

Alex. C. Humphreys, ... New York, N. Y.
Karl Eilers, ... New York, N. Y.
W. G. Miller, ... Toronto, Can.

Directors.

(To serve for three years.)

R. W. Raymond, ... New York, N. Y.
Charles H. Snow, ... New York, N. Y.
Theodore Dwight, ... New York, N. Y.

[Secretary's Note.—The complete list of all officers of the Institute will be found on p. x. of this volume. The following explanation, first published in *Bi-Monthly Bulletin*, No. 8, March, 1906, p. viii., is here repeated in order to recall to old members, and convey to new ones, the relations of the two governing bodies as determined by the Certificate of Incorporation of the Institute, and the Constitution and By-Laws adopted in accordance therewith.

The body legally responsible for the business management is the Board of nine Directors (three elected annually to serve three years), which elects its own officers. This body, for reasons of practical convenience, is composed of well-known members residing in New York City, and able to attend, without serious inconvenience or expense, the necessary meetings of the Board. The officers of this Board are legally the officers of the Institute. But, apart from business management, the Board exercises no control over the election of members, or the professional and technical work of the Institute, except that its vote is required to elect honorary members, upon the recommendation of the Council.

The Council is a body constituted in all respects (except that it has no Treas-

urer) like the Council existing before the incorporation of the Institute, in January, 1905, and charged with all duties and powers, except those which the Board of Directors must legally perform. It elects members, appoints the times and places of professional meetings, and controls the publication and distribution of papers and volumes, etc. Its members (President, Vice-Presidents and Councilors) are elected by the members of the Institute, voting in person or by proxy, and after publication of the nominations received; and it is intended to represent, as far as practicable, both the professional and the geographical distribution of the membership. Consequently, whatever professional honor attaches to official position belongs to membership in the Council, rather than in the legal Board of Directors. This remark implies no disparagement of the members of the latter body, every one of whom has served, or is now serving, as a member of the Council. But it is only fair to explain that their election and continued re-election as Directors is simply a matter of legal convenience.]

Proposed Amendments to the Constitution.

Action upon the following amendments to the Constitution, proposed at the Annual Meeting, Feb. 18, 1908, was postponed to the next business meeting:

To Art. II. After the first sentence of this Article, add the following sentence: "These classes may be sub-divided by the Council, according to profession, length of membership, nationality, or other conditions not inconsistent with the provisions of this article."

To Art. III. In the first line, substitute "fifteen" for "ten" dollars.

PROCEEDINGS OF THE BOARD OF DIRECTORS.

The following acts of the Directors are reported for the information of members:

At a meeting held Jan. 5, 1909, after recommendation by the Council, By-Law XIV. was unanimously amended by striking out the word "Bi-Monthly" in the first and second sentences thereof, the amended By-Law reading:

XIV. *Publications.*

The publications of the Institute shall include a periodical called the *Bulletin* of the American Institute of Mining Engineers, which shall contain reports of proceedings, professional papers, notices and other matters of interest to members. From the annual dues paid by each Member or Associate, five dollars shall be deducted and applied as a subscription to the *Bulletin* for the year covered by such payment.

At a meeting held Jan. 5, 1909, after unanimous recommendation by the Council, the following were unanimously elected Honorary Members of the American Institute of Mining Engineers:

Prof. Richard Beck, of Freiberg, Saxony, in recognition of his distinguished service to technical literature, and the cordial acknowledgment and co-operation which he has given to American investigators in economic geology.

Emil Schroedter, of Düsseldorf, Germany, in recognition of his distinguished service to metallurgical science and industry, as Secretary and Director of the German Society of Iron-Masters, and as technical editor of *Stahl und Eisen*, and also of the cordial hospitality and assistance which he has so long and so often generously rendered to the Institute and its members.

The following was unanimously elected an Honorary Associate of the American Institute of Mining Engineers:

James M. Swank, in recognition of his long and distinguished service as historian and statistician of the American iron and steel industry.

At a meeting held Jan. 5, 1909, Mr. E. E. Olcott was unanimously re-elected a Trustee of the United Engineering Society, to serve for a second term of three years.

At a meeting held Jan. 5, 1909, Theodore Dwight, Treasurer of the Land Fund Committee, submitted a report of the work accomplished up to Jan. 1, 1909, of which the following brief summary is published:

LAND MORTGAGE.

Original obligation on account of one-third equity in land, 25-33 West 39th Street, New York, N. Y., Engineering Societies Building, covered by mortgage,	$188,000.00
Paid on the original principal to date,	63,000.00
Balance at 4 per cent. per annum ($5,000),	$125,000.00

LAND FUND.

Subscriptions to date,		$69,950.00
Paid on original principal,	$63,000.00	
Reimbursed American Institute of Mining Engineers for interest paid from its regular income,	6,400.00	
Balance in National Bank of Commerce (non-interest drawing),	550.00	
		$69,950.00

Of the amount so far collected, about $47,000 has been contributed by eleven members and firms.

At a meeting held Feb. 16, 1909, directly after the adjournment of the annual business meeting of the Institute, the following officers were re-elected for the ensuing year:

President, James Gayley; *Vice-President,* James Douglas; *Secretary,* R. W. Raymond; and *Treasurer,* Frank Lyman.

Financial Statement.

The following statement of receipts and expenditures from Jan. 1 to Dec. 31, 1908, is published by authority of the Board of Directors:

RECEIPTS.

Balance from statement of January, 1908,		$4,306.75
Annual dues,*	$39,130.32	
Life memberships,	900.00	
Initiation fees,	2,490.23	
Binding of *Transactions*,	3,764.13	
Sale of publications, electrotypes, advertising, and miscellaneous receipts,	7,788.71	
		54,073.39
Interest on bonds and deposits,		1,219.88
		$59,600.02

* $19,415 of this amount has been applied to subscriptions to the *Bi-Monthly Bulletin* in accordance with post-office regulations.

DISBURSEMENTS.

Printing Vol. XXXVIII. of the *Transactions*, *Bi-Monthly Bulletin*, extra pamphlets, and advertising expenses, etc.,	$11,859.60	
Printing circulars and ballots,	130.00	
Binding Vol. XXXVIII. of the *Transactions*,	3,458.00	
Binding miscellaneous volumes,	183.03	
Engraving and electrotyping,	672.67	
Secretary's department, including clerks, stenographers, and expenses of editing and proof-reading,	10,196.00	
Treasurer's department, including collection of dues, shipping, etc.,	5,586.00	
Librarian and assistants,	1,718.13	
Postage,	3,812.42	
Stationery,	512.77	
Express and freight charges,	1,250.17	
Telephone,	196.15	
Telegrams, cables, carfares, etc.,	36.44	
Office supplies and repairs,	116.58	
Refunding over-payments,	19.51	
Insurance premiums (Surety),	95.00	
Collection charges,	47.96	
Extra clerical assistance,	120.00	
Special stenographers and expenses of meetings,	1,664.57	
Auditing,	125.00	
Office cleaning and sundry expenses,	33.22	
		$41,833.22
Interest at 4 per cent. on $130,000 principal of land mortgage on 25 to 33 West 39th St.,	5,200.00	
Quota of current expenses of building 25 to 33 West 39th St.,	6,700.00	
		$11,900.00
Special editing, part payments on printing and binding special edition, Index Vols. I. to XXXV., and new volume of Genesis of Ore-Deposits,		289.55
Library additions of books, periodicals, etc., binding of exchanges, and stationery (expenditure from appropriation of $1,000),		945.29
Furniture, etc., balance of $2,000 appropriation for furniture, etc., fitting up new offices,	480.61	
Sundry fittings and incidentals,	262.77	
		743.38
Balance,		3,888.58
		$59,600.02

NEW YORK, N. Y., February 8, 1909.

We have examined the above statement, compared it with the books and vouchers and find same correct.

(Signed) BARROW, WADE, GUTHRIE & Co.,
Certified Public Accountants.

REPORT OF THE COUNCIL FOR THE YEAR 1908.

MEETINGS.

Two meetings of the Institute were held during the year 1908 for the reading and discussion of papers—the Ninety-fourth Meeting, in New York, Feb. 18 to 22, and the Ninety-fifth Meeting, in Chattanooga, Oct. 1 to 7.

A detailed record of the proceedings of these meetings, including a description of the entertainments and excursions connected therewith, has been published and duly distributed to the members: the New York meeting in *Bi-Monthly Bulletin*, No. 20, March, 1908, pp. 239 to 288, and the Chattanooga meeting in *Bi-Monthly Bulletin*, No. 24, November, 1908, pp. 1187 to 1213. At the New York meeting there were presented 29 papers and 9 discussions, oral and written; in these discussions 26 separate contributors participated. At the Chattanooga meeting there were presented 36 papers and 13 discussions, oral or written; in these discussions 33 contributors participated. At the New York meeting the names of 182 members and guests were registered at the Institute headquarters; this number, however, does not represent all who were present at the sessions and excursions. At the Chattanooga meeting the number of members and guests registered or attending, in whole or in part, the various excursions around Chattanooga, or the train-trip to Copperhill and Ducktown, amounted to 232, which doubtless does not include all who attended the sessions or excursions.

PUBLICATIONS.

Transactions.—Volume XXXVIII. of the *Transactions*, an octavo of 1,025 pages, comprising 53 papers and 7 discussions presented during the year 1907, was issued and distributed to members in June, a few weeks earlier in the year than the corresponding appearance of Volume XXXVII. With the exception of the index, now being compiled, the material for Volume XXXIX., forming in all about 1,000 pages, is in the hands of the printer, and it is expected that the bound volume will be off the press and ready for distribution in June, 1909.

Bi-Monthly Bulletin.—Six numbers of the *Bi-Monthly Bulletin* (Nos. 19 to 24) have been published and distributed promptly throughout the year 1908. These numbers contain the technical papers and discussions of the Institute (in "subject to revision" form), and announcements of general interest to the members of the Institute, such as Library accessions and requirements during the year 1908; a list of the serial publications in the libraries of the American Institute of Electrical Engineers, the American Society of Mechanical Engineers, and the American Institute of Mining Engineers, now available for consultation by members of the Institute; lists of proposed members and associates; changes of address; deaths of members; obituary notices; Report of the United Engineering Society for the year 1907; Index of Subjects and Authors, etc. The number of pages occupied by technical papers and discussions amounts to 1,214, to which are to be added 324 pages of announcements, and 92 pages of advertising matter, making a total of 1,630 pages of printed matter for the year.

The editorial and business management of the *Bi-Monthly Bulletin*, Volume XXXVIII. and the forthcoming Volume XXXIX. of the *Transactions* continues in charge of Dr. Joseph Struthers, Assistant Secretary and Editor of the Institute.

MEMBERSHIP.

Changes in membership have taken place during the year as follows: 227 members and 15 associates have been elected; 1 member has been reinstated and 5 associates have become members; the deaths of 1 honorary member, 47 members and 1 associate have been reported; 60 members and 6 associates have resigned; and 78 members and 1 associate have been dropped from the roll by reason of non-payment of dues, loss of correct address, etc.* These changes are shown in the accompanying table.

The total membership on Jan. 1, 1909, was 4,241, as compared with 4,192 on Jan. 1, 1908—a net gain for the year of 49 members.

* Many of these, no doubt, will be reinstated, as has been the case in former years.

Membership of the American Institute of Mining Engineers, Jan. 1, 1909.

	Honorary Members.	Members.	Associates.	Totals.
Membership Dec. 31, 1907	12	4,018	162	4,192
Gains: By Election	....	227	15	242
Change of Status	....	5	....	5
Reinstatement	....	1	....	1
Re-election	....	0	....	0
Losses: By Resignation	....	60	6	66
Change of Status	....	....	5	5
Dropping	....	78	1	79
Death	1	47	1	49
Total gains	....	233	15	248
Total losses	1	185	13	199
Membership Dec. 31, 1908	11	4,066	164	4,241

The list of deaths reported during the year 1908 comprises the following names, the figures in parentheses indicating the year in which the persons named were elected to membership:

Members and Associates.—Ai Arthur Abbott (1895), Peter T. Austen (1905), William Beals, Jr., (1897), Louis W. Bond (1906), Edward E. Brown (1881), Edward D. Chester (1890), William T. Climo (1906), L. U. Colbath (1887), John T. Conner (1900), Sterling B. Cox (1901), George Davey (1891), Pat Doyle (1879), Adolf Ekman (1902), Robert J. Forsythe (1896), Francis T. Freeland (1885), James B. Gallagher (1900), Oliver S. Garretson (1898), T. R. Gue (1889), James D. Hague (1903), Frederick S. Harris (1897), Max Heberlein (1904), Albert Helms (1889), Alfred E. Jessup (1898), Ralph I. Johnson (1902), Tom C. King (1906), Clermont Livingston (1905), Jawood Lukens (1884), Hoyt S. McComb (1906), John McConnell (1903), Donald R. Morgan (1905), E. G. N. North (1906), M. N. Srinivas Rao (1907), Francis A. Roepper (1903), Edward F. Schaefer (1907), Herbert P. Seale (1901), William Seaton, Jr. (1905), Fred. J. Shaler (1905), Harry L. Shrom (1901), Egbert Smit (1902), Henry Stern (1901), Charles J. Steffens (1907), Ernst A. Thies (1900), Herbert N. Tod (1902), Ralph W. Watson (1890), Hermann Wedding (Hon.), Cabell Whitehead (1890), John Wilkes (1883), Cary Wright (1900), William S. Yeates (1895).

Of these, James D. Hague has been made the subject of a special Biographical Notice (see *Trans.*, xxxix., frontispiece, and pp. 677 to 685).

MEMBERSHIP.

The following list comprises the names of those persons elected as members, who duly accepted election during the year 1909. The marks used to designate the different classes of membership are: Life Member, **; Member, *; Associate Member, †. Heavy-faced type signifies Honorary Membership.

*Roy H. Allen, ... Lunenburg, Mass.
*William G. Anderson, ... San Francisco, Cal.
†Johan O. W. Applegren, ... Chicago, Ill.
*Daniel McP. Armstead, ... Butte, Mont.
*Louis Baird, ... Etzatlan, Jalisco, Mex.
*Egbert H. Ballard, ... Everett, Mass.
*Howland Bancroft, ... Washington, D. C.
*John Barry, ... Animas Forks, Colo.
Richard Beck, ... Freiberg, Saxony, Germany.
*Henry T. Beckwith, ... Philadelphia, Pa.
*Charles N. Bell, ... Denver, Colo.
*Thomas H. Bentley, ... Socorro, N. M.
†Haakon A. Berg, ... Midland, Pa.
*James C. Besley, ... Hermosillo, Sonora, Mex.
*Robert M. Black, ... Berwind, W. Va.
*Fred H. Bostwick, ... Denver, Colo.
*Adam A. Boyd, ... Broken Hill, N. S. W., Australia.
*Albert H. Boyd, ... Denver, Colo.
*Dudley H. Bradlee, Jr., ... Santa Barbara, Chihuahua, Mex.
*D. Owen Brooke, ... Birdsboro, Pa.
*David R. C. Brown, ... Aspen, Colo.
*Donald J. Browne, ... Rossland, British Columbia, Can.
*Henry L. Browne, ... Swansea, Ariz.
*Ernest F. Burchard, ... Washington, D. C.
*Herbert T. Burls, ... London, England.
*Alfred P. Busey, Jr., ... Campo Seco, Cal.
*John M. Cairns, ... Glasgow, Scotland.
†James F. Calbreath, Jr., ... Denver, Colo.
*Cipriano R. Careaga, ... Bilbao, Spain.
*Samuel W. Cohen, ... Cobalt, Ontario, Can.
*Walter B. Cole, ... Salt Lake City, Utah.
*Edwin J. Collins, ... Duluth, Minn.
*Alexander J. F. Crauford, ... Silver City, N. M.
*Erle V. Daveler, ... Tonopah, Nev.
*John Deegan, ... Los Angeles, Cal.
*Frank W. DeWolf, ... Urbana, Ill.
*Horace C. Dickey, ... Columbia, Pa.
*Edmund S. Dickinson, ... Florence, Wis.
*Wallace F. Disbrow, ... Hazel Green, Wis.
*Charles W. Dodge, Jr., ... Denver, Colo.
*Bernhard Draeger, ... Lubeck, Germany.
*James Duffin, ... Cananea, Sonora, Mex.
*Raoul G. Dufourcq, ... Arizpe, Sonora, Mex.

*Herbert L. Eaton, ... Bob, Nev.
*George S. Evans, ... Silver City, N. M.
*George Fairbairn, ... Celaya, Guanajuato, Mex.
*Raffaele Farina, ... London, England.
*John R. Finletter, ... Globe, Ariz.
*William B. Foote, ... Geneva, N. Y.
*William J. Forbach, ... Casa Grande, Ariz.
*Carlton E. Fortney, ... Coahuila, Mex.
*L. H. French, ... New Rochelle, N. Y.
*Jokichi Fujioka, ... Chikugo, Japan.
*John T. Fuller, ... Murfreesboro, Ark.
*Toranosuke Furukawa, ... Tokyo, Japan.
*Henry F. E. Gamm, ... Bayonne, N. J.
*John W. Gankroger, ... Indé, Durango, Mex.
*John G. G. George, ... Nacozari, Sonora, Mex.
*Frederick C. Gilbert, ... Durango, Colo.
*George H. Glover, Jr., ... New York, N. Y.
*William E. Gordon-Firebrace, ... London, England.
*Paul Grammel, ... Sumatra, Dutch East Indies.
*John Greenall, ... Allentown, Pa.
*George G. Greene, ... Washington, D. C.
*Charles C. Gressang, ... Wright, W. Va.
*Charles H. Grill, ... Bob, Nev.
*Carl E. Grunsky, ... Poto, Peru.
*Raimund T. Guernsey, ... Easton, Pa.
†Harold O. Hammond, ... New York, N. Y.
*James H. Hance, ... Salt Lake City, Utah.
*Fred G. Hawley, ... Cananea, Sonora, Mex.
*Nathaniel Herz, ... New Haven, Conn.
*Frank R. Hewitt, ... Asheville, N. C.
*Edwin Higgins, ... New York, N. Y.
*Roger R. Hill, ... Saginaw, Mich.
*Cleaveland Hilson, ... Electric, Mont.
*Ole G. Hoaas, ... Wardner, Idaho.
*Frederick C. Holmes, ... Santiago de Cuba, Cuba.
*Clarence T. Hopkins, ... Jerome, Ariz.
*Richard J. Horschitz, ... Haileybury, Ontario, Can.
*John J. Howard, ... Wharton, N. J.
*Ernest Howe, ... Newport, R. I.
*Edward C. Hugon, ... Hautes Pyrénées, France.
*Archabald S. Hummell, ... High Bridge, N. J.
*Frank W. Iredell, ... New York, N. Y.
*David D. Irwin, ... Nyack, N. Y.
*Herbert E. Jackman, ... Cobalt, Ontario, Can.
*Norman L. Jenks, ... Butte, Mont.
*H. Seaver Jones, ... Oxford, N. J.
*Irving S. Josephs, ... New York, N. Y.
*Tatsuzo Kamiyama, ... Shitsukigun, Okayamaken, Japan.
*Percy Kenyon, ... Concepcion del Oro, Zacatecas, Mex.
†Harold C. Kingsbury, ... Brisbane, Queensland, Australia.
*Harry E. Kirk, ... Cananea, Sonora, Mex.
*George Kislingbury, ... Los Angeles, Cal.

*Howard W. Kitson,	Colorado Springs, Colo.
*Warren L. Kluttz,	Thomas, Ala.
*Leslie M. Kozminsky,	New York, N. Y.
*S. Arthur Krom,	Plainfield, N. J.
*Henry M. Lancaster,	Wallace, Idaho.
*Seth S. Langley,	Indé, Durango, Mex.
*Robert G. Lassiter,	Virgilina, Va.
*Vaughan M. Lavery,	Goyllarisquisga, Peru.
*William A. Laycock,	London, England.
*Charles N. Lindley,	New York, N. Y.
*Harold B. Litchman,	University, Ala.
*James E. Little,	Steelton, Pa.
*Robert Livingstone,	Edmonton, Alberta, Can.
*Alexander Longwell,	Toronto, Ontario, Can.
*Dorsey A. Lyon,	Heroult, Cal.
*Eugene McAuliffe,	Chicago, Ill.
*Samuel W. McCallie,	Atlanta, Ga.
*Robert McCart, Jr.,	Indê, Durango, Mex.
*James F. McCarthy,	Wallace, Idaho.
*Atholl F. McEwen,	Cobalt, Ontario, Can.
*Milton H. McLean,	Morenci, Ariz.
*William B. McPherson,	Los Angeles, Cal.
*James H. Macia,	Tombstone, Ariz.
*Arthur W. Martin,	Carrizal, Chile.
*Manuel G. Masias,	Lima, Peru.
†Paul H. Mayer,	Cambridge, Mass.
*Charles Mentzel,	Copperhill, Tenn.
*David G. Miller,	Denver, Colo.
*Frank S. Mills,	Andover, Mass.
*Charles A. Moffett,	Birmingham, Ala.
*Ledlie D. Moore,	Santiago de Cuba, Cuba.
*Henry A. Morin,	Gowganda, Ontario, Can.
*Duncan M. Munro,	Hostotipaquillo, Jalisco, Mex.
*Robert Musgrave,	Yzabal, Sonora, Mex.
*Roy V. Myers,	Knoxville, Tenn.
*Foster Naething,	Rico, Colo.
*Aquila C. Nebeker,	Topliff, Utah.
*James Negus,	Boppy Mountain, N. S. W., Australia.
*Andrew W. Newberry,	Kelvin, Ariz.
†Roger W. Newberry,	New Haven, Conn.
*John O. Norbom,	Berkeley, Cal.
*Bartolomé Novoa,	Lima, Peru.
*Albert D. Oberly,	Scottdale, Pa.
*Abner A. Osborn,	Parryville, Pa.
*Morris B. Parker,	El Paso, Texas.
*Horace F. Parsons,	Garfield, Utah.
*William B. Patrick,	Rico, Colo.
†Karl A. Pauly,	Schenectady, N. Y.
*Francis A. Paz-Soldan,	Lima, Peru.
*Marmaduke Peckitt,	Wharton, N. J.
*John G. Perry,	Denver, Colo.
*William M. Peschel,	Perth Amboy, N. J.

*Jesse C. Porter,	Brooklyn, N. Y.
Alexandre Pourcel,	Paris, France.
*John W. Powell,	Coleman, Alberta, Can.
†Benjamin S. Pray,	Boston, Mass.
*Philip P. Reece,	Hartshorne, Okla.
*Thomas V. Reeves,	Alameda, Cal.
*G. Clinton Ripley,	Craft, Cal.
*Levi E. Riter, Jr.,	Salt Lake City, Utah.
*John T. Roberts, Jr.,	Buffalo, N. Y.
*Joseph H. Rodgers,	Seattle, Wash.
†Stephen Royce,	Cambridge, Mass.
*Guy H. Ruggles,	Santa Rita, N. M.
*Branch E. Russell,	Nacozari, Sonora, Mex.
*Chard O. Sanford,	Goldroad, Ariz.
*Mortimer F. Sayre,	Bisbee, Ariz.
*Charles H. Schlacks,	San Francisco, Cal.
*Herbert G. Schader,	Oatman, Cal.
*Gustave Schrader,	Sutter Creek, Cal.
*Harold Schroder,	Port Kembla, N. S. W., Australia.
Emil Schroedter,	Düsseldorf, Germany.
*Joseph S. Shaw,	Algoma, W. Va.
*Charles F. Shelby,	Cananea, Sonora, Mex.
*George M. Shoemaker,	Princeton, W. Va.
*Michael J. Slattery,	San Martin Hidalgo, Jalisco, Mex.
*Eugene A. Smith,	University, Ala.
†Henry De Witt Smith,	New London, Conn.
*Edgar K. Soper,	Ithaca, N. Y.
*Daniel M. Stackhouse,	Johnstown, Pa.
*Chevalier B. Staples,	Cranbrook, British Columbia, Can.
*Donald Steel,	Brownsville, Cal.
*Edward L. Stenger,	Pittsburg, Pa.
*Jesse A. Stewart,	McKinley, Minn.
*William C. Stratton,	Scottdale, Pa.
*Joseph S. Stringham,	Detroit, Mich.
*Guy C. Stoltz,	Mineville, N. Y.
*Archibald M. Strong,	Bishop, Cal.
James M. Swank (Associate),	Philadelphia, Pa.
*Charles D. Thompson,	New Haven, Conn.
*James B. Torbert,	Jersey Shore, Pa.
*Nicholas Truschkoff,	Jekaterinburg, Dept. of Perm, Russia.
*Tsok Kai Tse,	Wickenburg, Ariz.
*Fremont N. Turgeon,	Firmeza, Cuba.
*Albert A. Turner,	Beatty, Nev.
*W. A. Underhill,	New Market, Tenn.
*Lawrence H. Underwood,	Wheeling, W. Va.
*Percy A. Wagner,	Johannesburg, South Africa.
*William L. Walker,	Salt Lake City, Utah.
*George H. Warren,	Minneapolis, Minn.
*Chester W. Washburne,	Washington, D. C.
*William Wearne,	Hibbing, Minn.
*Walter S. Weeks,	Cambridge, Mass.
*Robert A. Weiss,	New York, N. Y.

*Rush J. White, ... Wallace, Idaho.
*Bernard A. Wilkinson, ... Mexico City, Mex.
*A. Emory Wishon, ... Fresno, Cal.
*Albert G. Wolf, ... Telluride, Colo.
*John G. Worth, ... Denver, Colo.
*Edward T. Wright, ... Chicago, Ill.
*Manji Yoshimura, ... Marunouchi, Tokyo, Japan.
*Clinton M. Young, ... Lawrence, Kan.
*Richard Ziesing, ... Cleveland, Ohio.
*Frank P. Zoch, ... Cold Spring-on-the-Hudson, N. Y.

Deaths.

The following list comprises the names of members whose deaths have been reported to the Secretary of the Institute during the year 1909:

Date of Election.	Name.	Date of Decease.
1905.	*William Adams,	December 8, 1909.
1906.	**Robert S. Brooks,	July 6, 1909.
1905.	*James W. Brown,	October 23, 1909.
1895.	*Raymond B. Brown,	March 30, 1909.
1882.	*F. G. Bulkley,	December 28, 1908.
1878.	**Charles B. Dudley,	December 21, 1909.
1888.	*Richard Eames, Jr.,	December 15, 1909.
1880.	*George G. Francis,	———— —, ———.
1871.	**Persifor Frazer,	April 7, 1909.
1904.	*Edward L. Fuller,	January 29, 1909.
1908.	*John M. Grice,	September 15, 1909.
1896.	*Rasmus Hanson,	September 28, 1909.
1903.	*Harold H. Harvey,	August 24, 1909.
1888.	*Alphonse Hennin,	August 18, 1908.
1883.	*H. August Hunicke,	April 5, 1909.
1899.	*Algernon K. Johnston,	October 3, 1909.
1876.	*Jerome Keeley,	July 4, 1909.
1891.	*Emil Krabler	October 24, 1909.
1878.	*L. G. Laureau,	June 6, 1909.
1903.	*Frank A. Lucy,	June 7, 1909.
1897.	*John E. McCurdy,	December 15, 1908.
1881.	*Charles C. Mattes,	June 8, 1909.
1875.	*William Metcalf,	December 5, 1909.
1904.	*Stephen C. Miller,	December 8, 1908.
1875.	††Israel W. Morris,	December 18, 1909.
1896.	*John W. Nesmith,	December 17, 1909.
1900.	*Bertel Peterson,	February 10, 1909.
1881.	*Robert Pitcairn,	July 25, 1909.
1900.	*Jasper R. Rand,	March 30, 1909.
1905.	*Davis Richardson,	April 5, 1909.
1873.	*William H. Singer,	September 4, 1909.
1906.	*James Stirling,	June 26, 1909.
1903.	*Delos V. A. Williams,	May 15, 1909.
1871.	**Thomas F. Witherbee,	July 11, 1909.

* Member. ** Life Member. †† Life Associate.

Proceedings of the Ninety-Sixth Meeting, New Haven, Conn., February, 1909.

LOCAL COMMITTEES.

GENERAL RECEPTION COMMITTEE.—Louis Valentine Pirsson, *Chairman;* John Duer Irving, *Secretary;* Frank L. Bigelow, *Treasurer;* Thomas G. Bennett, Arthur H. Day, Alton Farrel, Samuel Higgins, Frederick J. Kingsbury, Jr., John K. Punderford, Henry B. Sargent, Calvert Townley, Eli Whitney, Charles F. Brooker, Henry Brewer, George T. Surface, Edward S. Dana, Augustus J. DuBois, Frank A. Gooch, Herbert E. Gregory, Louis D. Huntoon, Charles B. Richards, Horace L. Wells, David Daggett, J. Arnold Norcross, William E. Ford, Joseph W. Roe.

ENTERTAINMENT.—L. V. Pirsson, *Chairman;* C. B. Richards, H. L. Wells, A. J. DuBois, Henry B. Sargent, Thomas G. Bennett, David Daggett.

FINANCE.—Frank L. Bigelow, *Chairman;* Henry B. Sargent, Calvert Townley, Henry Brewer, Samuel Higgins, Eli Whitney.

INFORMATION.—J. D. Irving, *Chairman;* W. E. Ford, H. L. Wells, L. V. Pirsson, George T. Surface.

EXCURSIONS.—L. D. Huntoon, *Chairman;* Arthur H. Day, Alton Farrel, Frank L. Bigelow, John K. Punderford, H. E. Gregory, Frederick J. Kingsbury, Jr., C. F. Brooker, Joseph W. Roe, J. Arnold Norcross.

The first session, held Tuesday evening, Feb. 23, in North Sheffield Hall, was called to order by Louis V. Pirsson, Chairman of the Local Committee, who introduced Prof. Russell H. Chittenden, Dean of the Sheffield Scientific School of Yale University. Professor Chittenden, in behalf of the Governing Board, felicitated the Institute on its visit to New Haven, and cordially welcomed the members and guests to the University and the city. Dr. James Douglas, who presided at the meeting, responded in behalf of the Institute.

The Secretary announced that, upon the proposal of many members, the unanimous report of the Committee on Membership, and the unanimous recommendation of the Council, the following members had been unanimously elected by the Board of Directors Honorary Members in the American Institute of Mining Engineers:

Prof. Richard Beck, of Freiberg, Saxony, in recognition of his distinguished service to technical literature, and the cordial

acknowledgment and co-operation which he has given to American investigators in economic geology.

Emil Schroedter, of Düsseldorf, Germany, in recognition of his distinguished service to metallurgical science and industry, as Secretary and Director of the German Society of Iron-Masters, and as technical editor of *Stahl und Eisen*, and also of the cordial hospitality and assistance which he has so long and so often generously rendered to the Institute and its members.

Similarly, the Board of Directors unanimously elected James M. Swank an Honorary Associate of the American Institute of Mining Engineers, in recognition of his long and distinguished service as historian and statistician of the American iron and steel industry.

The following papers were presented in oral abstract by the authors:

Conservation of Natural Resources, by James Douglas, New York, N. Y.

The Coal-Fields of the United States, by Marius R. Campbell and Edward W. Parker, Washington, D. C. (Presented by Mr. Parker.)

The Iron-Ore Supply of the United States, by C. Willard Hayes, Washington, D. C.

The second session, held in the same place on Wednesday morning, Feb. 24, was called to order by past-President Robert H. Richards, who presided.

The following paper, which had been freely distributed to all members and associates, and an extra edition of 1,750 to others specially interested in the subject, was called for discussion:

* A Sea-Level Canal at Panama: A Study of Its Desirability and Feasibility, by Henry G. Granger, Cartagena, Colombia, S. A.[1]

In the absence of the authors, written discussions were presented in oral abstract by the Secretary from the following participants: Gustav Schwab, New York, N. Y.; John C. Oakes, Galveston, Tex.; A. Woodroffe Manton, New York, N. Y. This paper was also discussed orally by W. L. Saunders,

* Distributed in printed form.

[1] *Bulletin* No. 25, January, 1909, pp. 1 to 37. Not included in this volume, at the request of the author.

New York, N. Y.; Charles Whiting Baker, New York, N. Y.; J. D. Evans, Lowell, Mass.; and Mr. Granger.[2]

The third session, held at the same place Wednesday afternoon, Feb. 24, was called to order by past-President Robert H. Richards, who presided.

The following paper was presented in oral abstract by the author:

The Mining-Course at the Sheffield Scientific School, Yale University, by John D. Irving, New Haven, Conn.[3] (Discussion orally by James F. Kemp, New York, N. Y.; R. W. Raymond, New York, N. Y.; Howard Eckfeldt, So. Bethlehem, Pa.; A. J. Hoskin, Denver, Colo.; and H. O. Hofman, Boston, Mass.[3])

The following paper, illustrated by lantern-views, was presented in oral abstract by the author:

The Hammond Mining and Metallurgical Laboratory of the Sheffield Scientific School, Yale University, by Louis D. Huntoon, New Haven, Conn.

The following paper, illustrated by lantern-views, was presented in oral abstract by Professor Roe:

Graphic Solution of Kutter's Formula, by L. I. Hewes and Joseph W. Roe, New Haven, Conn.

Discussion (additional) of the paper of Mr. Granger, A Sea-Level Canal at Panama, by Henry G. Granger, Cartagena, Colombia, S. A.[2]

The fourth session, held at the same place Wednesday evening, was called to order by past-President Robert H. Richards, who presided.

The following papers, illustrated by lantern-views, were presented in oral abstract by the authors:

* Driving Headings in Rock-Tunnels, by William L. Saunders, New York, N. Y.

* Vanadium-Deposits in Peru, by D. Foster Hewett, Pittsburg, Pa. (Discussion by James F. Kemp, New York, N. Y., and A. A. Blow,[3] New York, N. Y.)

---

* Distributed in printed form.

[2] *Bulletin* No. 31, July, 1909, pp. 623 to 666. Not included in this volume.

[3] Not furnished for publication.

The fifth and final session, held at the same place Thursday morning, Feb. 25, was called to order by Vice-President W. L. Saunders, who presided.

In the absence of the author, the following paper, accompanied by lantern-views, was presented in oral abstract by Edward W. Parker:

Coal-Mines and Plant of the Stag Cañon Fuel Co., by Jo. E. Sheridan, Silver City, N. M.

In addition to those papers marked with an asterisk (*), already mentioned, the following papers were distributed:

* The Mineral Wealth of America, by R. W. Raymond and W. R. Ingalls, New York, N. Y.[4]

* Hydraulic Dredging for Gold-Bearing Gravels, by Henry G. Granger, Cartagena, Colombia, S. A.

* Blast-Pressure at the Tuyeres and Inside the Furnace, by R. H. Sweetser, Columbus, Ohio.

* Pressure-Fans *vs.* Exhaust-Fans, by Audley H. Stow, Maybeury, W. Va.

* Discussion of the paper of J. J. Rutledge, The Clinton Iron-Ore Deposits in Stone Valley, Huntingdon County, Pa., by William Kelly, Vulcan, Mich., and H. S. Chamberlain, Chattanooga, Tenn.

* Biographical Notice of James Duncan Hague, by R. W. Raymond, New York, N. Y.[5]

* Discussion of the paper of George B. Lee, The Corrosion of Water-Jackets of Copper Blast-Furnaces, by J. A. Thomson, Pullman, Wash.[6]

* Discussion of the paper of William H. Shockley, The Bogoslovsk Mining Estate, by H. W. Mussen, Collingwood, Ont., Canada.[7]

* Discussion of the paper of Dr. R. W. Raymond, Dip and Pitch, by H. L. Smyth, Cambridge, Mass.[8]

* Discussion of the paper of H. M. Chance, A New Theory of the Genesis of Brown Hematite-Ores; and a New Source of Sulphur Supply, by Charles Catlett, Staunton, Va.[9]

---

* Distributed in printed form.

[4] *Bulletin* No. 27, March, 1909, pp. 249 to 264. Not included in this volume.

[5] *Trans.*, xxxix., frontispiece, and pp. 677 to 685.

[6] *Ibid.*, pp. 815 to 817.

[7] *Ibid.*, pp. 897 to 898.

[8] *Ibid.*, pp. 913 to 916.

[9] *Ibid.*, pp. 916 to 920.

* Discussion of the paper of Frank Firmstone, An Unusual Blast-Furnace Product; and Nickel in Some Virginia Iron-Ores, by John J. Porter, Cincinnati, Ohio.[10]

The following papers in manuscript form were available for consultation by members desiring discussion:

The American Institute of Mining Engineers and the Conservation of Mineral Resources, by John Birkinbine, Philadelphia, Pa.

A Reliable Steel Rail and How to Make It, by James E. York, New York, N. Y.

Pan-Amalgamation: an Instructive Laboratory-Experiment, by H. O. Hofman and C. R. Hayward, Boston, Mass.

The Residual Brown Iron-Ores of Cuba, by C. M. Weld, New York, N. Y.

Development in the Size and Shape of Blast-Furnaces in the Lehigh Valley, as shown by the Furnaces at the Glendon Iron Works, by Frank Firmstone, Easton, Pa.

The Laws of Fissures, by Blamey Stevens, Seattle, Wash.

Kentucky Fluorspar, and Its Value to the Iron and Steel Industries, by F. Julius Fohs, Lexington, Ky.

The Concentration of Silver-Lead Ores at the Works of Block 10 Co., Broken Hill, N. S. W., Australia, by V. T. Stanley Low, Broken Hill, Australia.[11]

The Treatment of Slime on Vanners, by Rudolf Gahl, Morenci, Ariz.

Metal-Losses in Copper-Slags, by Lewis T. Wright, San Francisco, Cal.

Biographical Notice of Herman Wedding, by Emil Schroedter, Düsseldorf, Germany. (Translation by Dr. R. W. Raymond.)

Discussion of the paper of John Hays Hammond, Professional Ethics, by Henry Louis, Newcastle, England.

Discussion of the paper of Carl Scholz, Effect of Humidity on Mine-Explosions, by Howard N. Eavenson, Gary, W. Va., and A. T. Shurick, Washoe, Mont.

---

* Distributed in printed form.

[10] *Trans.*, xxxix., p. 921.

[11] *Bulletin* No. 33, September, 1909, pp. 763 to 793, and in more extensive form in the *Australian Mining Standard*, Mar. 31, Apr. 7, 14, and 21, 1909. Not included in this volume.

Discussion of the paper of Albert F. J. Bordeaux, The Silver-Mines of Mexico, by Alfred H. Bromly, Mexico City, Mexico, and Albert F. J. Bordeaux, Thonon les Bains, France.

Discussion (additional) of the paper of Henry G. Granger, A Sea-Level Canal at Panama, by L. M. Haupt, Philadelphia, Pa.; R. R. Hancock, New York, N. Y.; H. L. Millner, Washington, D. C., and Ernest Howe, Newport, R. I.[12]

The following paper was read by title for future publication:

A Rational Basis for the Conservation of Mineral Resources, by Joseph A. Holmes, Washington, D. C.[13]

Excursions and Entertainments.

An account of the visits and entertainments of the New Haven meeting of the Institute, and the excursions to Ansonia and Bridgeport, prepared through the courtesy of Prof. Louis D. Huntoon, of the Local Committee, was printed in *Bulletin* No. 28, April, 1909, pp. 430 to 436.

List of Members and Guests Registered at New Haven or Attending the Excursions to Ansonia and Bridgeport.

(Doubtless incomplete.)

Alling, J. W., New Haven, Conn.
Baker, Charles Whiting, New York, N.Y.
Bancroft, Howland, Washington, D. C.
Barnum, G. S., New Haven, Conn.
Bassett, W. H., Torrington, Conn.
Benham, F. N., Bridgeport, Conn.
Benham, Mrs. F. N., Bridgeport, Conn.
Benham, F., Jr., Bridgeport, Conn.
Bennett, T. G., New Haven, Conn.
Bennett, Mrs. T. G., New Haven, Conn.
Bigelow, F. L., New Haven, Conn.
Blackwell, A. S., Mandalay, Burma.
Blow, A. A., New York, N. Y.
Blow, George, New York, N. Y.
Brewer, Henry, New Haven, Conn.
Brewster, F. F., New Haven, Conn.
Bristol, E. S., New Haven, Conn.
Brooker, C. F., New Haven, Conn.
Brown, R. W., New York, N. Y.
Bushnell, W. G., New Haven, Conn.
Chapman, George, New Haven, Conn.
Chittenden, R. H., New Haven, Conn.
Clements, J. Morgan, New York, N. Y.
Cogswell, W. B., Syracuse, N. Y.
Cogswell, Mrs. W. B., Syracuse, N. Y.
Crawford, W. H., Nashville, Tenn.
Daggett, David, New Haven, Conn.
Dana, R. T., New York, N. Y.
Day, A. H., New Haven, Conn.
Day, David T., Washington, D. C.
Douglas, James, New York, N. Y.
Du Bois, A. J., New Haven, Conn.
Du Bois, H. W., Philadelphia, Pa.
Dwight, Theodore, New York, N. Y.
Eckfeldt, H., So. Bethlehem, Pa.
Emmons, S. F., Washington, D. C.
English, Mrs. B. K., New Haven, Conn.
English, H. F., New Haven, Conn.
English, James, New Haven, Conn.
English, L. H., New Haven, Conn.
Evans, J. D., Lowell, Mass.
Farrel, Alton, New Haven, Conn.
Forbes, R. R., New York, N. Y.
Ford, W. E., New Haven, Conn.
Ford, J. M., New Haven, Conn.
George, J. G., Watertown, N. Y.

[12] *Bulletin* No. 31, July, 1909, pp. 623 to 666. Not included in this volume.
[13] *Idem*, No. 29, May, 1909, pp. 469 to 476. Not included in this volume.

Godley, G. McM., New York, N. Y.
Gooch, F. A., New Haven, Conn.
Granger, A. O., New York, N. Y.
Granger, H. G., New York, N. Y.
Grant, C. M., New York, N. Y.
Graton, Louis C., Washington, D. C.
Gregory, H. E., New Haven, Conn.
Hansell, N. V., New York, N. Y.
Hayes, C. Willard, Washington, D. C.
Herz, N., New Haven, Conn.
Hewett, D. Foster, Pittsburg, Pa.
Higgins, Samuel, New Haven, Conn.
Hofman, H. O., Boston, Mass.
Holm, J. D., Stillwater, Minn.
Hooker, T., New Haven, Conn.
Hoskin, A. J., Golden, Colo.
Hotchkiss, Henry, New Haven, Conn.
Hotchkiss, Stuart, New Haven, Conn.
Howe, Ernest, Newport, R. I.
Howes, D. W., Stone Ridge, N. Y.
Huntoon, L. D., New Haven, Conn.
Huntoon, Mrs. L. D., New Haven, Conn.
Hutchinson, E. S., Newton, Pa.
Ingalls, W. R., New York, N. Y.
Ingalls, Mrs. W. R., New York, N. Y.
Irving, J. D., New Haven, Conn.
Irving, Miss, New Haven, Conn.
Irwin, D. D., Boston, Mass.
Kemp, James F., New York, N. Y.
King, R. P., Atlanta, Ga.
Kingsbury, F. J., Jr., New Haven, Conn.
Kissam, W. A., Manila, P. I.
Lane, J. S., Brooklyn, N. Y.
Leckie, R. G., Sudbury, Can.
Ledoux, Albert R., New York, N. Y.
LeFevre, S., Mineville, N. Y.
LeWald, E. A., Tannersville, N. Y.
Lindgren, W., Washington, D. C.
Loudon, A., Norwalk, Conn.
Lynn, Thomas, New Haven, Conn.
McKenna, Charles F., New York, N. Y.
Maignen, P. S., Philadelphia, Pa.
Matcham, C. A., Allentown, Pa.
Matcham, Mrs. C. A., Allentown, Pa.
Matcham, Miss Dorothy, Allentown, Pa.
Matlack, E. V., St. Louis, Mo.
Matlack, Mrs. E. V., St. Louis, Mo.
Miller, J. W., New York, N. Y.
Morse, B. K., New York, N. Y.
Norcross, J. Arnold, New Haven, Conn.
Norris, R. V., Wilkes-Barre, Pa.
Ormrod, George, Emaus, Pa.
Ormrod, J. D., Emaus, Pa.
Orrok, G. A., New York, N. Y.
Otis, T. E., New York, N. Y.
Paine, T. Ward, New Haven, Conn.
Parker, Edward W., Washington, D. C.
Parsons, L. A., New York, N. Y.
Pirsson, L. V., New Haven, Conn.
Pirsson, Mrs. L. V., New Haven, Conn.
Pitman, S. M., Providence, R. I.
Pitman, Mrs. S. M., Providence, R. I.
Punderford, J. K., New Haven, Conn.
Raymond, Dr. R. W., New York, N. Y.
Richards, C. B., New Haven, Conn.
Richards, Robert H., Boston, Mass.
Ridgway, Robert, Poughkeepsie, N. Y.
Roe, J. W., New Haven, Conn.
Root, Edward, New Haven, Conn.
Sanborn, James F., Poughkeepsie, N. Y.
Sargent, H. B., New Haven, Conn.
Saunders, W. L., New York, N. Y.
Saunders, Miss, New York, N. Y.
Shipp, E. M., Newburgh, N. Y.
Smith, H. D., San Francisco, Cal.
Spalding, W. A., New Haven, Conn.
Sterrett, Douglas B., Washington, D. C.
Stoek, H. H., Scranton, Pa.
Storrs, L. S., Springfield, Mass.
Struthers, Dr. Joseph, New York, N. Y.
Surface, G. T., New Haven, Conn.
Thompson, C. D., Honesdale, Pa.
Thompson, S. F., Jenkintown, Pa.
Torrey, H. G., New York, N. Y.
Torrey, Mrs. H. G., New York, N. Y.
Townley, Calvert, New Haven, Conn.
Trowbridge, R., New Haven, Conn.
Wagner, H. G., New Haven, Conn.
Wallace, Thomas, New Haven, Conn.
Watson, C. B., Parkersburg, W. Va.
Wells, H. L., New Haven, Conn.
Wells, Mrs. H. L., New Haven, Conn.
Whitney, Eli, New Haven, Conn.
Wiley, Charles, New York, N. Y.
Wiley, William H., New York, N. Y.
Woodbury, F. C., Milwaukee, Wis.
York, James E., New York, N. Y.
York, Mrs. James E., New York, N. Y.
York, S. A., New Haven, Conn.

Proceedings of the Ninety-Seventh Meeting, Spokane, Wash., September, 1909.

COMMITTEES.

BUTTE, MONT.—Charles W. Goodale, *Chairman;* B. H. Dunshee, Benjamin B. Thayer, John Gillie, William Scallon, C. F. Kelly, H. A. Gallway, A. C. Carson, A. H. Wethey, John C. Adams, Oscar Rohn.

ANACONDA, MONT.—E. P. Mathewson, *Chairman;* William Wraith, J. C. Guinness.

CŒUR D'ALENE DISTRICT, IDAHO.—Roy H. Clarke, *Chairman;* Frederick Burbidge, Stanley Easton, J. F. McCarthy, H. L. Day, J. V. Richards, D. L. Huntington, J. B. Fisken.

SPOKANE, WASH.—E. J. Roberts, *Chairman;* L. K. Armstrong, *Secretary;* Charles P. Robbins, C. M. Fassett, J. C. Ralston, J. C. Haas, J. V. Richards, Roy H. Clarke, W. C. Miller, R. Marsh, J. M. Porter. Also, Western Branch of the Canadian Mining Institute, Thomas Kiddie, *Chairman;* E. Jacobs, *Secretary;* W. Fleet Robertson.

SEATTLE, WASH.—Frank A. Hill, *Chairman;* Chester F. Lee, *Secretary;* Milnor Roberts, M. K. Rodgers, C. M. Lewis.

TACOMA, WASH.—John H. Williams, *Chairman;* C. R. Claghorn, Henry Hewitt, Jr., Charles A. Foster, F. W. Clark, Roger Taylor, John N. Pott.

SALT LAKE CITY, UTAH.—Duncan MacVichie, *Chairman;* R. H. Bradford, J. M. Callow, Ellsworth Daggett, E. E. Nelson, R. S. Oliver, R. Forrester, C. C. Crimson, H. R. Ellis, Samuel Newhouse.

BINGHAM, UTAH.—D. C. Jackling, L. Hanchett, J. C. Dick, C. H. Doolittle, R. C. Gemmell, L. S. Cates, E. P. Jennings, M. M. Johnson, Samuel Newhouse, R. Forrester, G. Lavagnino, C. W. Saxman, B. F. Tibby, E. A. Wall, Thomas Weir, J. B. Risque, C. E. Allen, G. W. Metcalf, S. R. Woodbridge.

GARFIELD, UTAH.—A. J. Bettles, L. Hanchett, F. G. Janney, D. C. Jackling, Samuel Newhouse, R. Forrester, C. W. Whitley, P. B. Tracy, R. C. Gemmell.

BINGHAM JUNCTION AND MURRAY, UTAH.—C. W. Whitley, George W. Heintz, P. E. Barbour.

PARK CITY, UTAH.—George D. Blood, E. W. Durfee, F. W. Sherman, F. T. Williams.

TINTIC DISTRICT, UTAH.—P. J. Donohue, J. H. McChrystal, I. N. Dunyon, J. C. McChrystal, G. W. Riter, Col. C. E. Loose, C. E. Allen.

PUEBLO, COLO.—J. B. McKennan, F. E. Parks, H. A. Deuel, F. Guiterman, George A. Marsh, B. Hogarty, C. R. Rose, A. Stock.

The first session, held Tuesday afternoon, Sept. 28, 1909, in the beautiful Convention Hall of Masonic Temple, was called to order by Mr. E. J. Roberts, Chairman of the Spokane Local Committee, who introduced Mr. J. C. Ralston, City Engineer, representing the Mayor of Spokane and the Spokane Chamber

of Commerce. Mr. Ralston extended to the members and guests of the Institute a cordial welcome to the city of Spokane.

Mr. Ralston's address was acknowledged by President David W. Brunton, and responded to by the Secretary, Dr. R. W. Raymond.

The following paper was presented in oral abstract by the author:

* Modern Progress in Mining and Metallurgy in the Western United States, by David W. Brunton, Denver, Colo.

This paper was discussed by W. S. Ayres, Hazleton, Pa. (coal-recovery); Prof. William Kent, Sandusky, O. (debt of mining engineering to the mechanical engineer); Charles Catlett, Staunton, Va. (cost-keeping and efficiency-records); Dr. W. O. Snelling, Pittsburg, Pa. (explosives); Charles W. Goodale, Butte, Mont. (dust-recovery); Ernest Levy, Rossland, B. C. (replacement-mining); W. L. Saunders, New York, N. Y. (air-compression).

The second session of the Institute, held at the same place, Tuesday evening, Sept. 28, was called to order by President Brunton.

The following papers were presented in oral abstract by the authors:

Discussion (continued) of Mr. Brunton's paper, Modern Progress in Mining and Metallurgy in the Western United States, by Thomas Kiddie, Northport, Wash. (fume-condensation).

* Modern Practice of Ore-Sampling, by David W. Brunton, Denver, Colo.

In the absence of the author, the Secretary presented in oral abstract the following paper:

* Dust-Explosions in Coal-Mines, by Franklin Bache, Fort Smith, Ark. Discussed by Dr. R. W. Raymond.

The following paper was presented in oral abstract by the author:

Causes of Variation in Ore-Sampling, by Thomas Kiddie, Northport, Wash.[1]

---

* Distributed in printed form.

[1] *Quarterly Bulletin of the Canadian Mining Institute*, No. 8, pp. 47 to 56 (Dec., 1909). Not included in this volume.

Under a mutual agreement between the Councils of the American Institute of Mining Engineers and the Western Branch of the Canadian Mining Institute, it was decided to hold a joint session of both Institutes during the meeting at Spokane. This agreement also provided that each of the two societies should be free to publish such papers and discussions, presented at this session, as it should deem desirable, and accordingly the text of the papers presented by the Western Branch of the Canadian Mining Institute will be found either in this volume, or in the *Journal of the Canadian Mining Institute.*

The third and concluding session of the Institute was selected as best suited for the joint meeting.

This session, held at the same place, Wednesday morning, Sept. 29, was presided over by President Brunton, of the American Institute of Mining Engineers, and President Kiddie, of the Western Branch of the Canadian Mining Institute.

The following papers were presented in oral abstract by the authors:

* The Ruble Hydraulic Elevator, by J. McD. Porter, Spokane, Wash.

The Conservation of Coal in the United States, by E. W. Parker, Washington, D. C. Discussed by Dr. R. W. Raymond,[2] New York, N. Y., and Prof. William Kent,[2] Sandusky, Ohio.

Coal-Mining in Southeastern British Columbia and in Alberta, by E. Jacobs, Victoria, B. C.[2]

In the absence of the author, the following paper was presented in oral abstract by Mr. Jacobs:

The Galt Coal-Field, Lethbridge, Alberta, by W. D. L. Hardie, Lethbridge, Alberta, Canada. Discussed by W. S. Ayres, Hazleton, Pa. (Galt and Bankhead coal-fields)[2]; Milnor Roberts, Seattle, Wash. (Nicola coal-region); Wm. Fleet Robertson, Victoria, B. C. (North Vancouver Island and Queen Charlotte Island fields)[2]; Charles Catlett, Staunton, Va. (coal-prices)[2]; Frederick Keffer, Greenwood, B. C. (coal- and coke-prices at furnaces)[2]; and Thomas Kiddie, Northport, Wash. (general).[2]

In addition to the papers already noted, the following were read by title for future publication:

---

* Distributed in printed form.

[2] Not furnished for publication.

* The Formation and Enrichment of Ore-Bearing Veins, by George J. Bancroft, Denver, Colo. (supplementary paper).

* Need of Instrumental Surveying in Practical Geology, by Benjamin Smith Lyman, Philadelphia, Pa.

* The Assay and Valuation of Gold Bullion, by Frederic P. Dewey, Washington, D. C.

The Cyaniding of Silver-Ores in Mexico, by Albert F. J. Bordeaux, Thonon les Bains, France.

* The Ventilating-System of the Comstock Mines, Nevada, by George J. Young, Reno, Nev.[3]

An Adjustable Pyrometer-Stand, by L. W. Bahney, Palo Alto, Cal.

Glass Mine-Models, by Edmund D. North, Tonopah, Nev.

* Postscript to paper, The Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes, by H. O. Hofman and W. Mostowitsch, Boston, Mass.

Cyaniding Slime, by Mark R. Lamb, Milwaukee, Wis.

* Influence of Ingot-Size on the Degree of Segregation in Steel Ingots, by Henry M. Howe, New York, N. Y.

Preparing and Recording Samples for Use in Technical Assay-Laboratories, by Louis D. Huntoon, New Haven, Conn.

The Geology, Mining, and Preparation of Barite in Washington County, Missouri, by A. A. Steel, Fayetteville, Ark.

Discussion of the paper of Charles R. Keyes, Genesis of the Lake Valley, New Mexico, Silver-Deposits, by William M. Courtis, Detroit, Mich., and Bernard MacDonald, Guanajuato, Mex.

Discussion of the paper of Henry M. Howe, Piping and Segregation in Steel Ingots, by P. H. Dudley, New York, N. Y.

Discussion of the paper of Charles R. Keyes, Ozark Lead- and Zinc-Deposits; Their Genesis, Localization, and Migration, by E. R. Buckley, Flat River, Mo.

Discussion of the paper of D. F. Hewett, Vanadium-Deposits in Peru, by James F. Kemp, New York, N. Y.

Discussion of the paper of Hofman and Hayward, Pan-Amalgamation; an Instructive Laboratory-Experiment, by E. A. H. Tays, San Blas, Sinaloa, Mexico.

* The Influence of Bismuth on Wire-Bar Copper, by H. N. Lawrie, Portland, Ore.

---

* Distributed in printed form.

[3] Not finally approved by author in time to include in this volume.

The Barometric and Temperature Conditions at the Time of Dust-Explosions in the Appalachian Coal-Mines by N. H. Mannakee, Williamson, W. Va.

Influence of Top-Lag on the Depth of the Pipe in Steel Ingots, by H. M. Howe, New York, N. Y.

* The Limit of Fuel-Economy in the Iron Blast-Furnace, by N. M. Langdon, Mancelona, Mich.

* Borax-Deposits in the United States, by Charles R. Keyes, Des Moines, Iowa.

* Conditions and Costs of Mining at the Braden Copper Mines, Chile, by William Braden, New York, N. Y.

Review of Modern Cyanide Practice in the United States and Mexico, by S. F. Shaw, Los Angeles, Cal.[4]

The Nicola Valley Coal-Field, British Columbia, by Milnor Roberts, Seattle, Wash.

A New Separator for the Removal of Slate from Coal, by W. S. Ayres, Hazleton, Pa.

Excursions and Entertainments.

An elaborately-illustrated description of the trip through the Yellowstone Park and the excursions to Butte, Anaconda, and the Cœur d'Alène district, and the subsequent excursions to Seattle, Tacoma, Portland and the Dalles of the Columbia, Salt Lake City, Bingham Cañon, Glenwood Springs, and through the Royal Gorge to Pueblo and Colorado Springs, was printed in *Bulletin* No. 36, December, 1909, pp. 1065 to 1118.

List of Members and Guests Constituting the Special Excursion Party.

Ayres, W. S., Hazleton, Pa.
Ayres, Mrs. W. S., Hazleton, Pa.
Bellinger, A. R., Syracuse, N. Y.
Bostwick, F. H., Denver, Colo.
Bostwick, Mrs. F. H., Denver, Colo.
Brooke, D. Owen, Birdsboro, Pa.
Brooke, Mrs. D. Owen, Birdsboro, Pa.
Brunton, D. W., Denver, Colo.
Catlett, Charles, Staunton, Va.
Chamberlain, H. S., Chattanooga, Tenn.
Chamberlain, Mrs., Chattanooga, Tenn.
Dougherty, J. W., Steelton, Pa.
Douglas, Miss J. L., Brooklyn, N. Y.
Fries, Miss Anna, Philadelphia, Pa.
Glendenning, Miss J. A., New York, N.Y.
Greenfield, T. B., London, Eng.
Harrington, Arthur, Philadelphia, Pa.
Harrington, Miss Helen, Phila., Pa.
Harrington, M. H., Philadelphia, Pa.
Harrington, Mrs. M. H., Phila., Pa.
Hutchinson, E. S., Newtown, Pa.
Hutchinson, Mrs. E. S., Newtown, Pa.

* Distributed in printed form.

[4] *Bulletin* No. 31, July, 1909, pp. 591 to 619. Not included in this volume.

Jones, T. D., Hazleton, Pa.
Kanda, Reiji, Tokyo, Japan.
Kelly, Wm., Vulcan, Mich.
Kelly, Mrs. Wm., Vulcan, Mich.
Kent, Wm., Sandusky, Ohio.
Kent, Mrs. Wm., Sandusky, Ohio.
Lawton, A. H., New York, N. Y.
Lilly, John, Lambertville, N. J.
Lilly, Mrs. John, Lambertville, N. J.
Lilly, Wm., Lambertville, N. J.
McCrery, Charles, Birmingham, Ala.
MacFarland, Mrs. F. L., Denver, Colo.
Mitchell, W. S , Haileybury, Ont., Can.
Mitchell, Mrs. W. S., Haileybury, Can.
Nesmith, J. W., Denver, Colo.
Nesmith, Mrs. J. W., Denver, Colo.
Parker, E. W., Washington, D. C.
Perry, J. G., Denver, Colo.
Perry, Mrs. J. G., Denver, Colo.
Pilling, Ross, Philadelphia, Pa.
Pilling, W. S., Philadelphia, Pa.
Pilling, Mrs. W. S., Philadelphia, Pa.
Pilling, Miss M. B., Philadelphia, Pa.
Pinkney, H. H., Macdonald, W. Va.
Pitkin, S. H., Cleveland, Ohio.
Raymond, Dr. R. W., New York, N. Y.
Saunders, Miss Emily, Philadelphia, Pa.
Saunders, Miss Jean, New York, N. Y.
Saunders, Miss Louise, New York, N. Y.
Saunders, Mrs. W. B., Philadelphia, Pa.
Saunders, W. L., New York, N. Y.
Saunders, W. L., Jr., Philadelphia, Pa.
Shurick, A. T., Washoe, Mont.
Smith, Mrs. T. B., Birdsboro, Pa.
Snelling, Dr. W. O., Pittsburg, Pa.
Steiger, George, Washington, D. C.
Struthers, Dr Joseph, New York, N. Y.
Vaughan, A. E., New York. N. Y.
Vaughan, Mrs. A. E., New York, N. Y.
Wellman, S. T., Cleveland, Ohio.
Weiss, C. R., Philadelphia, Pa.

List of Members and Guests in Attendance at Spokane and Other Cities Visited.

(Doubtless incomplete.)

Adair, James B., Seattle, Wash.
Adair, Mrs. James B., Seattle, Wash.
Adair, Miss Minor G., Seattle, Wash.
Adams, Mrs. J. C., Butte, Mont.
Adams, M. A., Seattle, Wash.
Adams, Wm. H., Seattle, Wash.
Adams, Wm. H., Jr., Seattle, Wash.
Adams, John C., Butte, Mont.
Allen, C. E., Bingham, Utah.
Anderson, James, Seattle, Wash.
Armstead, Mrs. D. M., Butte, Mont.
Armstrong, L. K., Spokane, Wash.
Austin, Leonard S., Salt Lake, Utah.
Austin, W. Lawrence, Riverside, Cal.
Auzias, Turenne R., Seattle, Wash.
Barbour, P. E., Murray, Utah.
Bennette, Nelson, Tacoma, Wash.
Bettles, A. J., Garfield, Utah.
Blair, A. F., Seattle, Wash.
Blood, George D., Park City, Utah.
Bogardus, C. E., Seattle, Wash.
Boykin, James C., Seattle, Wash.
Bradford, R. H., Salt Lake City, Utah.
Burbidge, Fred'k, Silver King, Idaho.
Caetani, Gelascio, Wardner, Idaho.
Caldwell, F. M., Seattle, Wash.
Callow, J. M., Salt Lake City, Utah.
Cambier, A., Pueblo, Colo.
Carroll, Eugene, Butte, Mont.
Carroll, Mrs. Eugene, Butte, Mont.
Carson, A. C., Butte, Mont.
Cates, L. S., Bingham, Utah.
Champion, John R., Leadville, Colo.
Claghorn, C. R., Tacoma, Wash.
Claghorn, Mrs. C. R., Tacoma, Wash.
Clarke, Roy H., Spokane, Wash.
Clark, F. W., Tacoma, Wash.
Cortwright, Miss A., Seattle, Wash.
Cozzens, Harmon, Pueblo, Colo.
Crimson, C. C., Salt Lake City, Utah.
Curtis, Miss Frank W., Seattle, Wash.
Daggett, Ellsworth, Salt Lake City, Utah.
Day, H. L., Spokane, Wash.
Deuel, H. A., Pueblo, Colo.
Dick, J. C., Bingham, Utah.
Donohue, P. J., Salt Lake, Utah.
Doolittle, C. H., Bingham, Utah.
Dunshee, B. H., Butte, Mont.
Dunshee, Mrs. B. H., Butte, Mont.
Dunyon, I. N., Salt Lake, Utah.
Durfee, E. W., Park City, Utah.
Easley, George R., Baker City, Ore.

Easton, Stanley, Wardner, Idaho.
Ellis, H. R., Salt Lake City, Utah.
Elmendorf, Mr.
Emmens, N. W., Trout Lake, B.C.,Can.
Emmons, C. D., Eugene, Ore.
Evans, George W., Seattle, Wash.
Evans, Mrs. George W., Seattle, Wash.
Falkenburg, M. T., Seattle, Wash.
Farmer, James L., Seattle, Wash.
Fassett, C. M., Spokane, Wash.
Fisken, J. B., Post Falls, Wash.
Forrester, R., Salt Lake City, Utah.
Foster, Charles A., Tacoma, Wash.
Gallway, H. A., Butte, Mont.
Gemmell, R. C., Bingham, Utah.
Gillie, John, Butte, Mont.
Gillie, Mrs. John, Butte, Mont.
Goodale, Charles W., Butte, Mont.
Goodsell, C. H., Spokane, Wash.
Griswold, C. T., Colorado Springs, Colo.
Guinness, J. C., Anaconda, Mont.
Guiterman, F., Pueblo, Colo.
Haas, J. C., Spokane, Wash.
Haas, Mrs. J. C., Spokane, Wash.
Hanchett, L., Bingham, Utah.
Heintz, George W., Murray, Utah.
Hewitt, Henry, Jr., Tacoma, Wash.
Hill, Frank A., Seattle, Wash.
Hill, Mrs. Frank A., Seattle, Wash.
Hilson, Cleaveland, Electric, Mont.
Hough, W. B., Wardner, Idaho.
Hodges, A. B. W., Grand Forks, Can.
Hodges, Mrs. A. B. W., Grand Forks, Can.
Hogarty, B., Pueblo, Colo.
Huntington, D. L., Post Falls, Wash.
Jackling, D. C., Bingham, Utah.
Jacobs, E., Victoria, B. C., Canada.
Jammes, George, Seattle, Wash.
Jammes, Mrs. George, Seattle, Wash.
Janney, F. G., Garfield, Utah.
Jenks, Mrs. A. W.
Jennings, E. F., Bingham, Utah.
Johnson, M. M., Bingham, Utah.
Keffer, Frederic, Greenwood, B.C., Can.
Keffer, Mrs. Fred., Greenwood, B.C., Can.
Kelly, C. F., Butte, Mont.
Kelly, Mrs. C. F., Butte, Mont.
Kennedy, N. H., Spokane, Wash.
Kiddie, Thomas, Northport, Wash.
Knight, E. C., Seattle, Wash.
Knight, Mrs. E. C., Seattle, Wash.
Labarthe, J., Trail, B. C., Canada.
Labarthe, Mrs. J., Trail, B. C., Canada.
Landes, Mrs. Henry, Seattle, Wash.
Lavignino, G., Bingham, Utah.
Lee, Chester F., Seattle, Wash.
Lee, Mrs. Chester F., Seattle, Wash.
Levy, Ernest, Rossland, B. C., Canada.
Lewis, Clancy M., Seattle, Wash.
Lewis, Mrs. Clancy M., Seattle, Wash.
Loose, Col. C. E., Salt Lake, Utah.
Luckenbel, J. C., Spokane, Wash.
McCarthy, J. F., Burke, Idaho.
McCaustland, E. J., Seattle, Wash.
McCaustland, Mrs. E. J., Seattle, Wash.
McChrystal, J. C., Salt Lake, Utah.
McClelland, J., Salt Lake, Utah.
McCrea, W. S., Spokane, Wash.
McKennan, J. B., Pueblo, Colo.
MacVichie, Duncan, Salt Lake, Utah.
Magnussen, C. E., Seattle, Wash.
Mannheim, P. A. L., New York, N. Y.
Marsh, R., Spokane, Wash.
Marsh, George A., Pueblo, Colo.
Mathewson, E. P., Anaconda, Mont.
Merrill, Mr., Salt Lake, Utah.
Metcalf, G. W., Bingham, Utah.
Miller, W. C., Spokane, Wash.
Moss, Milton, Huntsville, Ala.
Munson, H. C., Spokane, Wash.
Moore, Mrs. C. H., Butte, Mont.
Morony, Mrs. J. G., Butte, Mont.
Nelson, E. E., Salt Lake City, Utah.
Nevin, Charles, Butte, Mont.
Newhouse, Samuel, Salt Lake City, Utah.
Oliver, R. S., Salt Lake City, Utah.
Parks, F. E., Pueblo, Colo.
Phœnix, C. E., Tacoma, Wash.
Phœnix, Mrs. C. E., Tacoma, Wash.
Porter, J. McD., Spokane, Wash.
Pott, John N., Tacoma, Wash.
Pott, Mrs. John N., Tacoma, Wash.
Raht, August, Denver, Colo.
Rankin, W. J., Jr., Seattle, Wash.
Ralston, J. C., Spokane, Wash.
Ralston, Mrs. J. C., Spokane, Wash.
Richards, J. V., Spokane, Wash.
Richardson, S. H., Republic, Wash.
Richardson, Mrs. S.H., Republic, Wash.
Risque, J. B., Bingham, Utah.
Riter, G. W., Salt Lake, Utah.
Robbins, Charles P., Spokane, Wash.
Roberts, E. J., Spokane, Wash.
Roberts, Milnor, Seattle, Wash.

Roberts, Miss Milnora, Seattle, Wash.
Robertson, Wm. Fleet, Victoria, Can.
Robertson, Mrs. W. Fleet, Victoria, Can.
Robertson, Douglas, Victoria, B.C., Can.
Rodgers, M. K., Seattle, Wash.
Rodgers, Master, Seattle, Wash.
Rohn, Oscar, Butte, Mont.
Rohn, Mrs. Oscar, Butte, Mont.
Rose, C. R., Pueblo, Colo.
Sanders, Wilbur E., Los Angeles, Cal.
Saxman, C. W., Bingham, Utah.
Scallon, William, Butte, Mont.
Sherman, F. W., Park City, Utah.
Smith, Joseph F., Salt Lake, Utah.
Stock, A., Pueblo, Colo.
Taylor, H. S., Chicago, Ill.
Taylor, Roger, Tacoma, Wash.
Thayer, Benjamin B., Butte, Mont.
Thomson, F. A., Seattle, Wash.
Tibby, B. F., Bingham, Utah.
Todd, Miss J., Seattle, Wash.
Tolman, L. P., Seattle, Wash.
Tracy, P. B., Garfield, Utah.
Trewarthe, James W. H., Victoria, Can.
Van Deusen, Mrs., Seattle, Wash.
Weston, Samuel P., Seattle, Wash.
Wall, E. A., Bingham, Utah.
Weir, Thomas, Bingham, Utah.
Wethey, A. H., Butte, Mont.
Wharton, J. R., Butte, Mont.
White, Richard M., Seattle, Wash.
Whitley, C. W., Garfield, Utah.
Williams, F. T., Park City, Utah.
Williams, John H., Tacoma, Wash.
Williams, Mrs. John H., Tacoma, Wash.
Williams, Percy, Vancouver, B. C.
Woodbridge, T. R., Bingham, Utah.
Wraith, William, Anaconda, Mont.

PAPERS.

Studies of Illinois Coals.*

(Chattanooga Meeting, October, 1908.)

TABLE OF CONTENTS.

I. INTRODUCTION.

BY H. FOSTER BAIN.†

THE recently aroused public interest in the conservation of our natural resources has peculiar importance to mining-men, since they deal with resources which are stored products. Within certain limits, the fertility of a worn-out soil may be restored, deforested areas can be replanted, one year's wasted water-supply is followed by another; but coal and ores, once taken out, cannot be mined again, and should therefore be conserved with special care. This means, however, not that mining should be restricted, but that it should be done most economically and with the minimum of waste.

---

* SECRETARY'S NOTE.—This paper has been prepared, under the general direction of Mr. Bain, by himself and the other gentlemen named in the Table of Contents.

† Director of the Illinois State Geological Survey.

As a first step, we should take stock of our reserves and study our methods of production. This is peculiarly true of coal, because of the intimate relations of fuel-supply and industrial supremacy, the speed with which our coal-fields are being exploited, and the large waste attending the mining and utilization of coal. In 1906, according to the U. S. Geological Survey, the value of the total mineral production of the United States was $1,902,517,565, of which $513,079,809 was derived from the coal-mines. In 1907, notwithstanding the unfavorable industrial conditions of the second half of that year, the coal-output of the country was 480,450,042 tons, valued at $614,831,549. It is impossible to determine exactly the waste of coal attending this production; but it is perhaps approximately accurate to say that, for each ton mined and marketed, another ton was lost in the processes of handling and preparation, or abandoned underground. Engineers know that the waste in burning the coal was even greater. In short, only a very small proportion of the energy residing in our coal-beds is utilized under present conditions.

Any effort to remedy these conditions must be based upon careful studies and much experimental work. The U. S. Geological Survey has taken the lead, so far as relates to the national coal-reserves; but no single organization can hope to do all the work, and the individual States must be prepared to take part in it. In Illinois a beginning has already been made. Through the Engineering Experiment Station of the University of Illinois and the Illinois State Geological Survey, various problems relating to the occurrence, production, and utilization of Illinois coals are being studied. In the papers which follow, some of the results of these investigations are given, supplemented by discussions of certain phases of the subject by Messrs. Rice and Bement, engineers especially familiar, through private practice, with the particular questions involved.

Illinois has a large interest in everything relating to coal. Though far behind Pennsylvania in present production, it ranks second among the States, with an output in 1907 of 51,317,146 tons, valued at $54,687,382. In the amount of its coal-reserves it undoubtedly outranks any Eastern or Central State, and its geographic position adds importance to the fact.

Illinois Coal as a Type.—The character and composition of

Illinois coals has been discussed in some detail by Prof. S. W. Parr,[1] who has especially pointed out the inert character of a large amount of the volatile matter present. Table I. is an average analysis based on 24 analyses, or averages of analyses, of face-samples, made by the Illinois State Geological Survey. The individual analyses have been weighted in proportion to the production of the various counties, and all the figures are based on the coal as received, including mine-moisture and occluded gas. Detailed analyses are given on later pages.

TABLE I.—*Average Analysis of Illinois Coal.*

Moisture. Per Cent.	Volatile. Per Cent.	Fixed Carbon. Per Cent.	Ash. Per Cent.	Sulphur.[a] Per Cent.	Calorific Capacity. B.t.u.
12.60	35.99.	41.32	10.51	3.22	11,046

[a] Separately determined.

Table II. gives the composition of commercial deliveries in Chicago from a number of representative Illinois mines compared with face-samples. The figures of lump, mine-run and screenings represent in each case the average of a large tonnage, as actually delivered and sampled by the Fuel Engineering Co. For comparison, analyses of face-samples from the same mines are included in the table.

TABLE II.—*Average Composition of Illinois Coals.* (*Commercial Deliveries.*)

Samples.	Sulphur. Per Cent.	Moisture. Per Cent.	Dry Ash. Per Cent.	Cal. Cap. (Dry). B.t.u.	Number of Mines.
Face, ...	3.35	12.27	10.88	12,779	22
Lump, . . .	3.08	10.40	10.7	12,827	14
Mine-run, . .	3.10	11.60	15.50	11,990	14
Screenings, . .	3.90	13.80	19.10	11,319	19

In making comparisons it should be noted that both the ash and the B.t.u. values are calculated on a "dry-coal" basis. In later pages face-samples from various parts of the State are considered.

Briefly, all Illinois coals are bituminous, and, as contrasted with their principal market-competitors, are relatively high in sulphur, ash, moisture, and volatile matter. Moreover, as Professor Parr has pointed out, 40 per cent. of the volatile matter, or 14 per cent. of the whole coal, is non-combustible, as contrasted with 22 and 4.2 per cent. respectively, in the case of

[1] *Bulletin No. 3, Illinois State Geological Survey,* pp. 27 to 79 (1906).

Pocahontas, Va., coal, and 47 and 21.63 per cent. in North Dakota lignite. Illinois coals are essentially free-burning and non-coking. They are mainly used for heating and power-generation, and have no large or direct use in metallurgy. The amount of sulphur present precludes their use for furnace-coke and complicates the problem of storage. The large proportion of volatile matter introduces a smoke-problem when the coals are burned in cities, and the high content in ash also detracts from their value. Despite all these facts, they have a high average value for miscellaneous heating and for steam-generation, and many of them are excellently adapted for use in gas-producers. In a general way, it may be said that the Illinois-Indiana coals are not inherently as valuable as the coals of the Appalachian basin, but more valuable than those of the Michigan and Western Interior fields, excepting limited areas in western Arkansas and eastern Oklahoma.

Statement of the Problem.—To estimate the position of Illinois coals in the markets of the future, the following topics must be considered: (1) the distribution and amount of coal available in the field; (2) the quality of the coal; (3) mining-conditions and costs; (4) present methods of utilization; (5) possible future methods of utilization; and (6) the relations of the deposits to markets.

A complete discussion of all these factors is at present impracticable. In the series of papers here given the first has been briefly considered by F.W. DeWolf, Assistant State Geologist. Certain phases of the second are discussed by Messrs. Lindgren and Barker on the basis of work done under direction of Professor Parr for the State Survey and the Experiment Station. The third is discussed by G. S. Rice, Consulting Engineer. The fourth is discussed in two papers, the first of which, treating of the domestic consumption of coal, has been prepared by J. M. Snodgrass as a result of work being done under the direction of L. P. Breckenridge at the Engineering Experiment Station. The problem of burning Illinois coals without smoke, a most important one in such a consideration as the present, is taken up by A. Bement, Consulting Engineer. The possible future methods of utilization of our coals have some light shed on them through the discussion of the weathering of coal, by W. F. Wheeler, and the artificial

modification of coal at low temperatures, by C. K. Francis. Both papers are based upon work done under Professor Parr in the laboratories of the University of Illinois. Finally, I have pointed out in a general way the relations of the coal-field to the market.

While there are many obvious gaps in this symposium, it is hoped that the papers forming it will give information, important not only to producers and users of Illinois coals, but to many others as well.

II. The Coal-Resources of Illinois.

By Frank W. DeWolf.*

Geographic Relations.

Coal-bearing rocks underlie three-fourths of Illinois, including 85 of its 102 counties. The coal-area may be estimated at from 36,000 to 42,000 square miles—the largest area of bituminous coal within any single State. The unproductive part, as shown in Fig. 1, includes the northern one-fifth, a narrow belt bordering the Mississippi river, and a half-dozen small counties at the southern extremity.

Production and Reserves.

The production in 1907, according to the U. S. Geological Survey, was 51,317,146 tons, with a spot value of $54,687,382—the largest production so far reached, representing a gain of 23.7 per cent. over that of 1906. Illinois thus ranks second among coal-producing States, a position which it has held for 23 years, except in 1906, when West Virginia, on account of labor-conditions, ranked second. There are more than 400 shipping-mines scattered through 52 counties, and 33 other counties are probably underlain by coal, but as yet not commercially developed.

Considering the production of the State for the past 30 years, in five-year totals, shown in Table I., minor fluctuations are lessened and the rapid strides of increase are made prominent.

* Assistant State Geologist, Illinois State Geological Survey.

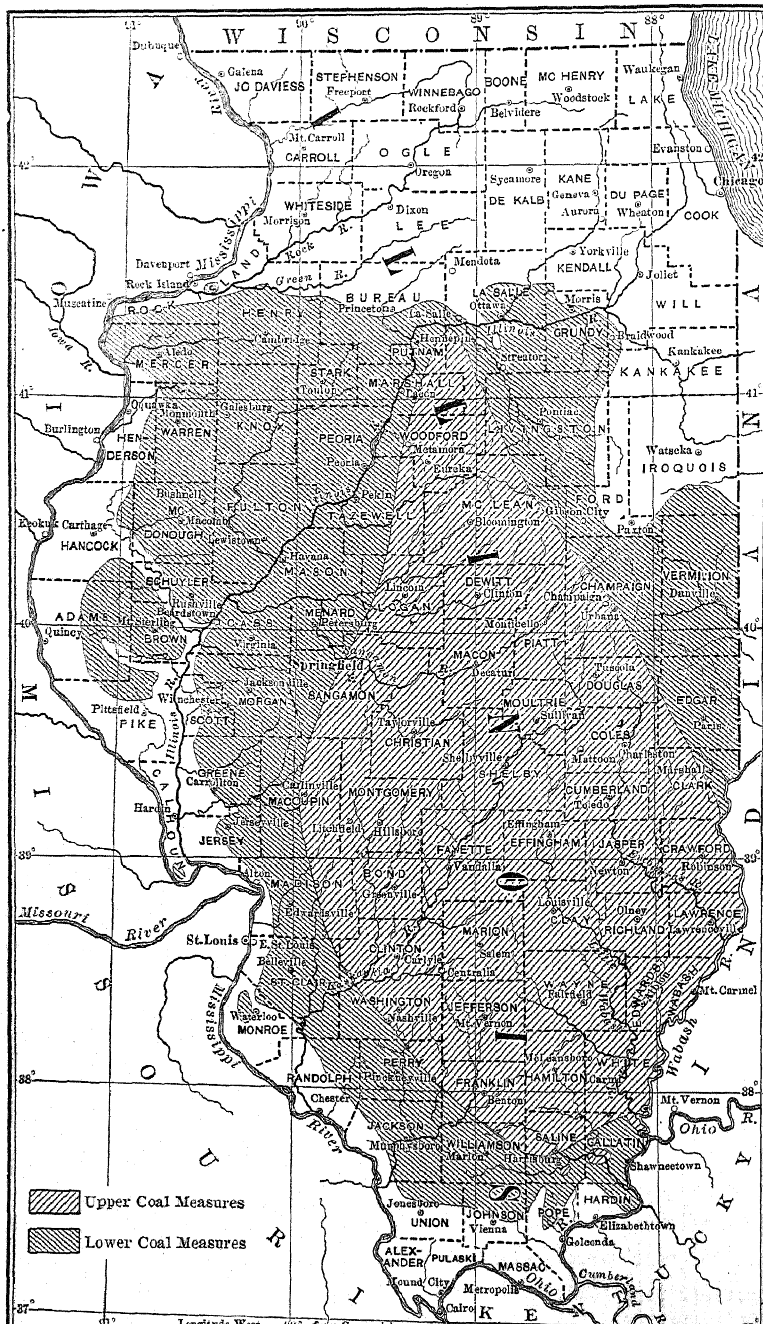


FIG. 1.—MAP OF ILLINOIS, SHOWING UPPER AND LOWER COAL MEASURES.

TABLE I.—*Production from* 1878 *to* 1907 *in 5-year Totals.*
(Round Numbers.)

Years.	Tons. Production.	Increase. Tons.	Increase. Per Cent.
1878–1882,	32,651,000	....	......
1883–1887,	59,764,000	27,113,000	83.0
1888–1892,	75,247,000	15,483,000	25.9
1893–1897,	78,377,000	3,130,000	4.1
1898–1902,	131,077,000	52,700,000	67.3
1903–1907.	204,646,000	73,569,000	56.1

While the total production has steadily increased, the percentage-rate of increase has, on the whole, diminished, and this has been interpreted by Parker, Fleming, and others to mean that, under present commercial tendencies, there will come, many years hence, for every producing State, a time when the rate of increase will be zero, and after which the total production will slowly diminish. Doubtless many factors will arise to modify the operation of this tendency, and in the case of Illinois the exhaustion of the coal-resources lies far in the future.

Estimates have recently been made of the total amount of coal originally under the State, and the amount still remaining. Such calculations are extremely uncertain; but, assuming the exploitation of all coal-beds 24 in. or more in thickness, and estimating according to present knowledge the thickness of each seam, the conclusions given in Table II. may be regarded as reasonable and prudent, though subject to revision.

TABLE II.—*Estimate of Illinois Coal-Resources.*
(Round Numbers.)

		Tons.
Original coal,		136,966,000,000
Mined to close of 1907,	645,868,309	
Wasted to close of 1907 (at 62 per cent. recovery),	245,429,957	
Mined and wasted to close of 1907,		891,000,000
Total reserves,		136,075,000,000

The largest area within which the amount of coal present is uncertain occupies the east central counties, where drill-records are scattered. A recent estimate by M. R. Campbell of the U. S. Geological Survey includes beds 20 in. and more in thickness, and places the original supply at 240,000,000,000 tons.

Geologic Relations.

The Illinois coal-region comprises about three-fourths of the Eastern Interior field, the remainder lying in neighboring parts of Indiana and Kentucky. It may once have been continuous with the Appalachian, Northern Interior, and Western Interior fields. There is great resemblance to the stratigraphy of the Indiana-Kentucky areas; and structurally the Eastern Interior basin is a unit. The geology of the coal-fields is now being studied in detail by the Illinois State and the U. S. Geological Surveys in co-operation. The data here given were obtained in the course of this study.

Structure.

Generalized cross-sections of the Illinois field, compiled by several geologists, show it to be spoon-shaped, the beds dipping gently towards a long axis which lies a short distance west of Lasalle and continues a little east of south to the southwest county of Indiana. The deepest part of the basin is in the vicinity of White county, and from here the strata rise more rapidly to the south than to the north, averaging over a considerable distance 40 ft., and locally 100 ft., per mile. The sides of the "spoon" show some minor longitudinal folds, notably the anticline which runs from Lasalle through the Illinois oil-field towards Princeton, Ind., a steep monocline at Duquoin, and a gentle anticline at Belleville. The southern margin of the basin shows numerous minor faults and at least one of consequence, which runs west and a little south from Shawneetown, and has here a down-throw to the north of more than 1,000 ft. The greater part of the basin contrasts with a narrow southern belt of rugged country, characterized by massive sandstones, but containing local areas of thick coal. Igneous dikes and other features along the southern margin of the basin indicate that the structure of the coal-field is in part related to the orogenic movements of southern Illinois and western Kentucky.

This structure has favored active mining around the edges of the basin, where the coal is most easily reached. Since, in the lowest area, the thick coal-beds lie 1,200 ft. or more below the surface, they will probably not be utilized there for some time.

Stratigraphy.

General Stratigraphy.—The rocks of the Coal Measures or Pennsylvanian series consist of alternating beds or lenses of shale and sandstone, with which are mingled thinner strata of limestone, coal, and fire-clay. There appear to be three general divisions of the rocks:

(1) The basal portion is composed chiefly of massive sandstones, and, according to David White, corresponds in age to the Pottsville formation of the Appalachian trough. This has a thickness of 650 ft. or more, in Johnson and Hardin counties, but diminishes rapidly in the west and north, being nearly or quite absent over much of the State. Coal No. 1 of the western counties lies near the top of this formation. Lower coals occur in southeastern Illinois and western Kentucky, and some of these were formerly mined.

(2) The second division extends from Coal No. 2 of the western and northern counties to Coal No. 6, or the Blue Band seam, and thus includes all the seams mined for shipment in the State. It is dominated by shale and contains a subordinate amount of sandstone. In age, it corresponds closely to the Allegheny formation or lower productive Measures of Pennsylvania, since, on the basis of plant-fossils, Coal No. 6 lies at or near the Upper Freeport, and No. 2 near the Kittanning horizon. This formation extends over nearly the whole coal-area, but its lower beds are not well known in the central part of the basin. At Peoria the total thickness is about 200 ft., and at Mattoon it appears to be 300 feet.

(3) The third and topmost division is dominated by shales, and contains no coal of present importance, though it is locally mined on a small scale. It occupies much of the coal-area, and reaches its greatest thickness (1,200 ft. or more) in the vicinity of Hamilton and White counties. From 275 to 350 ft. above its base occurs the Carlinville limestone, which, in the earlier State geological work, was accepted as a dividing-line between the Upper and Lower Coal Measures.

The total thickness of the Pennsylvanian rocks probably exceeds 2,000 ft., but around the edges of the basin much has been removed by erosion, and in a large part of the State the basal division is thin or absent. David White, who has contributed largely to recent studies, has shown that the earliest

beds were deposited in a restricted area in the southeastern counties, and that the favorable conditions for deposition of the Coal Measures gradually spread over the State, overlapping the eroded surface of the rocks, which are progressively older to the north.

The Coals.—In the work of early State surveys the Illinois coals were numbered from 1, at the bottom, to 16, at the top, and the same method was used in Kentucky, where, however, additional beds of the lower coals were found. The numbers, therefore, are confusing; Illinois Nos. 5 and 6 being identical respectively with Kentucky Nos. 9 and 11. Even in Illinois the numbers have been incorrectly assigned, and the same bed is now known under several numbers. Thus, Coal No. 7 of Saline and Williamson counties is undoubtedly the seam known as No. 6 and 7 at Duquoin, and as No. 6 in the Belleville region, and is probably the same as No. 6 at Peoria and No. 5 south of Springfield. The tracing of the Illinois coals is one of the interesting studies now in progress. Satisfactory work seems possible on the horizon of Coal No. 2, which White has found present from the northern long-wall district through the western belt of counties to Murphysboro, and also on the so-called "Blue Band" seam, called in different localities, as already remarked, Nos. 5, 6, and 7. Other beds of reasonable persistence will probably be found; but most of the Pennsylvania rocks seem to constitute "interfingering" lenses of comparatively local extent. There are at least three coal-seams of wide distribution, and from 3 to 9 ft. thick, besides others of local importance. Analyses and heat-values of Illinois coals are given on pp. 22 and 23.

Mining-Centers and Districts.

The State may perhaps be divided into natural districts on the basis of the varying fuel-value of the coals; and this study is now under way. The following notes, however, relate to important geographical districts or mining-centers recognized by the trade. The use of numbers does not imply correlation of the beds.

Williamson, Franklin, and Perry Counties.—Williamson county led the production of the State in 1907 with more than 5,500,000 tons, and its coal has a rapidly growing market.

No. 7, the Blue Band seam, which is from 5 to 10 ft. thick, averaging 9 ft. over a large area, is the greatest producer. The top-coal, about 20 in. thick, is frequently left to support the shale roof, and locally is withdrawn after the rooms have been mined out. The "blue band" is a clay or shale parting from 1 to 2 in. thick, and about 20 in. above the floor. There is a general northeast dip, amounting to 60 ft. per mile in the central part of the county. Local faults occur, sometimes with from 20 to 30 ft. displacement. The seam outcrops near Marion, and is absent from the southern part of the county, but elsewhere is reached by shafts, usually from 100 to 200 ft. deep. There is no sharp line between this field and its neighbors. The same seam is known in Perry and Franklin counties and in counties to the east. It maintains an approximate uniformity in physical character and thickness, but varies from place to place in fuel-value. At Duquoin on the west it is nearly horizontal, but on the east it dips rapidly and becomes thicker and somewhat better in quality. At Spillertown, another seam, 4 ft. thick, is mined 60 ft. below No. 7. This seam is probably equivalent to No. 5 of Saline county, and if we may judge from borings, may have a wide distribution in the Williamson county district.

Sangamon, Macoupin, Christian, Logan, and Macon Counties.—The Springfield district, extending into several adjoining counties, has long been one of the most important. Sangamon county produced more than 5,000,000 tons in 1907. The coal of the district is commonly known as No. 5, though recent work by Mr. Savage and me tends to confirm the suggestion made by Messrs. Bement,[1] Rice,[2] *et al.*, that there are probably two distinct beds mined in the district, No. 5 in the area north of Chatham and No. 6 south of that town. No. 5 is cut by numerous vertical clay veins from a few inches to 4 ft. in thickness, and lacks the "blue band" which characteristically occurs near the floor of No. 6. Both beds have a limestone cap-rock within a few feet of the coal. These coals are thought to be of the same age as Nos. 5 and 6 of the Peoria region, and the upper bed, No. 6, is probably the same as No. 6 of Belleville and No. 7 of Williamson county. No. 5 lies about 250 ft. below the surface in the vicinity of Springfield, at 425 ft. at Mount Olive

[1] *Bulletin* No. 3, *Illinois State Geological Survey*, p. 19 (1906).

[2] This volume, p. 31.

on the south, and at about 600 ft. at Decatur on the east. The average thickness of No. 5 is a little less than 6 ft. at Springfield, and about 4.5 ft. at Decatur, while No. 6 averages from 6 to 8 ft. in Macoupin county. There are three higher coals, all too thin to be mined at present, and lying respectively 50, 100, and 175 ft. above No. 5. There are likewise several coals below No. 5, but drilling has not been adequate to determine their commercial values. At Riverton to the east a diamond-drill record reports two seams, each measuring about 32 in., lying 125 and 250 ft. respectively below No. 5. There are also several other coals, which locally may develop into thick seams. A 4-ft. bed is reported to occur in this vicinity at a depth of 320 ft. below No. 5, but is known only from a churn-drill record.

St. Clair, Madison, Clinton, and Randolph Counties.—St. Clair county produced more than 4,500,000 tons in 1907. This district, known as the Belleville district, is not sharply set off from its neighbors, since the same coal-bed is mined under similar conditions in adjoining counties. It is again the Blue Band seam, with its parting near the base, and its limestone cap-rock, usually above the slate, but in some places directly overlying the coal itself. The thickness is from 5 to 7 ft. over much of this area, and the seam is reached by shafts from 100 to 300 ft. deep. It outcrops west of Belleville, and is eroded from the western part of the county. The general dip of the beds in St. Clair county, as demonstrated by the recent work of Dr. J. A. Udden, is eastward, from 10 to 20 ft. per mile. Local variations are frequent, and faults of 6 ft. displacement have been observed; but the general conditions are uniform. As to quality, analyses of face-samples indicate considerable irregular variation, so that no average can be given for the entire district. Borings indicate the presence of two deeper seams, one about 50 and the second from 100 to 150 ft. below No. 6; but their general workability has not been demonstrated.

Vermilion County.—During 1907 Vermilion county produced nearly 3,000,000 tons. It has long been an important area, shipping principally to the Chicago market. As described by M. R. Campbell,[3] there are three persistent coal-seams, two of

[3] *Danville Folio, U. S. Geological Survey* (1900).

which are worked. The top or Danville bed (No. 7) appears west of Vermilion river, and is mined along the outcrop and by shafts from 75 to 200 ft. deep. It is about 6 ft. thick around Danville, but more nearly 3 ft. ten miles further south. A band of bone or clay, lying from 6 to 20 in. above the floor, occurs in most of the sections. The Grape Creek coal (No. 6) lies from 20 to 80 ft. below the Danville, and is more important. It becomes thicker southward from Danville, and covers many square miles with a thickness of from 6 to 9 ft. A band of shale or sulphur frequently occurs about 2 ft. above the floor. Several borings have shown a seam from 185 to 220 ft. below the Grape Creek, and from 4 to 8 ft. thick, but badly broken by bands of shale and limestone.

Saline County.—Saline county is one of the newest and most rapidly growing producers. In 1907 its output was about 2,125,000 tons, a gain of 125 per cent. upon 1906. There are two seams, Nos. 7 and 5, underlying the northern two-thirds of this county and much of Gallatin on the east, each approximately 5 ft. thick, and lying from 90 to 150 ft. apart vertically. The upper is the Blue Band coal, which runs west into Williamson county and north into White and Hamilton. The lower seam is free from regular bands and has considerably higher heating-value, though in this respect the upper seam also is excellent. The seams outcrop to the south, and have a general northward dip of from 25 to 75 ft. per mile. Thus, the coal which outcrops at Equality, in Gallatin county, is from 900 to 1,000 ft. deep in Hamilton county, 25 miles north. Further northeast, diamond-drill records in the oil-fields indicate the presence of the same coals. An E–W. fault, with a downthrow to the north of more than 1,000 ft., crosses the middle of Saline and Gallatin counties, and is, perhaps, related to some minor faults and igneous intrusions in this district.

Fulton and Peoria Counties.—Fulton county produced more than 2,000,000 tons in 1907, and Peoria about half as much. No recent work has been done by the State Survey in Fulton, but Peoria has been studied carefully by Dr. J. A. Udden. The principal seam, called No. 5, is from 4 to 4.5 ft. thick, free from persistent partings, and dips gently SE., usually about 5 ft., and only locally as much as 60 ft., per mile. Shafts reach the coal at from 75 to 150 ft. In all, seven beds are present

here within 302 ft. of the surface, but only four have proved thick and persistent enough to be mined. No. 1, or the Lower Pottstown, is about 250 ft. below No. 5, and about 4 ft. thick, but is divided by a shale parting, 3 ft. thick, about 15 in. below the roof. This coal is no longer worked. No. 2 is about 130 ft. below No. 5 and 30 in. thick. It is worked by the long-wall method, and, according to the analysis of a mine-sample, appears to be a little better in quality than No. 5, though mining-conditions may render the commercial output inferior. No. 6 lies 70 ft. above No. 5, and has the characteristic band and roof-materials of the Blue Band seam, variously named Nos. 5, 6, and 7. The coal is a little less than 4 ft. thick, but lies near the surface, and has been locally faulted and broken, so as to render mining difficult.

Lasalle, Bureau, and Grundy Counties.—The Lasalle district includes three principal counties which together produce more than 5,000,000 tons. The largest production is obtained by long-wall mining from seam No. 2, or the "Third Vein." The coal averages about 3 ft. in thickness, and is blocky and of good quality. The method of mining introduces considerable ash in the screenings, and washers are used. The seam is reached by shafts from 125 to 450 ft. deep. About 140 ft. above No. 2 lies the seam, 4 ft. thick (or more), called No. 5 in former reports. About 40 ft. above it lies No. 7, which is extensively worked under the uplands of the region by room-and-pillar methods. The geological work of the present season should assist in the correlation of these upper beds with others of the State, and bring up to date our knowledge of this important field.

Western Field.—The counties along the western edge of the State are underlain by coals No. 1 and No. 2, recently traced by David White from Lasalle and Rock Island counties on the north to Murphysboro on the south. At present, mining in these counties is largely from No. 2, for local use. The lower seams usually measure from 2 to 3 ft. only, but the highlands contain areas of thicker upper coals also. In view of the present and future development of the clay industry of this district, the coal promises to be of great importance. The clay lies between No. 1 and No. 2, at the horizon of the famous Cheltenham clay of St. Louis, Mo.

III. The Sampling and Analysis of Illinois Coals.

By J. M. Lindgren.*

The importance of accurate sampling is evident. No matter how careful the analysis, the results are of little value if the sample be not truly representative. For example, if a sample of 100 lb. of coal, selected for quartering, contains a piece of pyrite larger than the pieces of coal, and weighing 3 lb., the inclusion of this pyrite in the sample would increase the ash by about 3 lb., and correspondingly raise the percentage of ash shown by subsequent analyses. Such a piece of pyrite should have been discarded as abnormal; yet, had it been of average size and presumably present in every other 100-lb. sample similarly taken, it would have been normal, and should not have been discarded. Another mistake is made in sampling when the best-looking piece of coal is selected as representative of the pile. Such samples are too frequently taken. Still another improper method is to select portions from different parts of the top of the pile, disregarding the coal underneath. This leads, in many cases, to serious errors, because the coal underneath is very likely to be of different character from that on the top.

Methods of Sampling.

Sampling Stock-Piles.—Probably the most common method of sampling coal is to select definite portions from different parts of the pile. In the case of a car-load of coal, sampled as it is unloaded, a good method is to select every twenty-fifth or thirtieth shovelful from different portions of the car. In sampling coal as it comes from the mine, it is customary to select a portion from each lot dumped into the coal-chutes.

W. F. Wheeler and Prof. S. W. Parr have invented a sampler which has given satisfactory results on coal of small size, and in the use of which it is unnecessary to handle the entire pile. It is made in two parts, one of which consists of a heavy, galvanized iron pipe, 6 ft. long and 8 in. in diameter, having at one end handles for revolving it, and at the other end, on opposite sides, two notches, slightly sharpened, so that, when the pipe is revolved, it will cut into the coal. The

* Assistant Chemist, Engineering Experiment Station of the University of Illinois.

second part is an iron rod, slightly longer than the pipe and sharpened at one end, to which end is securely fastened a pipe of thin galvanized iron 1 ft. by 7.5 in. in size, which will just fit into part No. 1. Inside this pipe is a series of sectional shutters, at right angles to the rod, which are arranged so as to fold fan-fashion when operated in one direction, but to unfold and close the opening when operated in the opposite direction through the turning of a lever-arm at the top of the iron rod. Part No. 2 also has handles for revolving. The apparatus is used as follows: Part No. 1 is shoved slightly into the coal, whereupon part No. 2 is inserted, and, by revolving, pushed further into the coal. When the interior pipe is filled, the bottom is closed by means of the lever, and the pipe is pulled out and emptied of its load of coal. Part No. 1 is then pushed still farther into the coal and a second portion is taken out; and so on, until the bottom of the pile is reached. By means of part No. 2 the coal is removed just in front of part No. 1, so that it can easily be advanced. Such a device, of course, can only operate on small sizes of coal.

Table I. gives ash-determinations, calculated to a dry-coal basis, of coal sampled in the three ways just described.

TABLE I.—*Dry Ash of Coal Sampled by Three Methods.*

Kind of Coal.	From Coal-Chutes. Per Cent.	From Cars. Per Cent.	From Bins By Pipe-Samples. Per Cent.
Sangamon egg,	17.87	16.63	17.46
Sangamon screenings,	17.13	17.01	17.22
Herrin egg,	14.32	14.90	14.32
Herrin screenings,	14.13	14.37	15.66
Westville egg,[a]	10.55	13.98	14.21
Westville screenings,	17.88	13.69	14.69

[a] In this case only the car-sample and the sample taken by pipe-sampler are comparable, because the sample from the chutes was picked clean at the top after each dump, after which a shovelful was selected for a sample, leaving a portion about 2 ft. thick, which was not cleaned. Of course, when the car was sampled during unloading this error was avoided.

Face-Sampling.—In taking a sample of coal which is to represent the quality of coal in the mine, it is extremely difficult to get a face of coal which is truly representative. Such a sample rarely represents the quality of coal actually mined, principally because of carelessness in mining. The results obtained at the St. Louis Exposition Fuel-Testing Plant showed

that the usual method of mine-sampling cannot be relied on to represent the average commercial product of the mine.[1] In many cases, however, it will correspond fairly with the lump coal.

A face-sample is taken in the following manner by the State Geological Survey: A face of coal, which represents as nearly as possible the average coal in the mine, is cleaned by taking off a layer of 2 or 3 in., after which all loose pieces are picked off the face and roof. A large piece of oil-cloth is then spread out on the floor to catch the coal as it is sampled. A strip of coal at least 5 lb. to the foot is cut down with the pick. Any bone, blue-band, or other impurity exceeding $\frac{3}{8}$ in. in thickness is discarded.

Quartering.—A sample of coal having been selected by any of the above stock-pile methods, is next reduced to an amount suitable for a working-sample in the laboratory. If it is in lumps, these should be broken up to about egg-size, pieces of pyrite, clay, etc., being removed and crushed, and then returned to the pile and the whole thoroughly mixed. After quartering, opposite quarters are kept; the remaining coal is crushed to about nut-size and again thoroughly mixed and quartered, and the opposite quarters, occupying the position of the ones which were not taken first, are selected. This method is continued until a sample of from 600 to 800 g., and of pea-size, is obtained. The sample is then ready for analysis. Face-samples, usually of smaller quantity than commercial samples, should be broken down to pass a screen of 0.5-in. mesh before quartering.

In order to determine to what extent a sample of coal could be quartered and still retain its original constitution with regard to ash, W. F. Wheeler conducted the following experiment: A 2-lb. sample of coal, obtained in the ordinary manner, was selected and quartered in the usual way, using a riffle for obtaining the working-sample for each division.

Fractional portions, representing $\frac{1}{4}$, $\frac{1}{8}$, $\frac{1}{16}$, $\frac{1}{32}$, $\frac{1}{64}$, and $\frac{1}{128}$ of this sample, were separately analyzed for ash. The results, exhibited in Table II., showed a marked accordance, the sample representing $\frac{1}{32}$ being the only exception.

Methods of Keeping Samples.

Face-samples are shipped to the laboratory from the mine in cylindrical tin cans, conforming to the U. S. Geological Survey

[1] *Professional Paper No.* 48, *U. S. Geological Survey*, p. 142 (1906).

TABLE II.—*Distribution of Ash in Coal during Quartering Two-Pound Sample with Riffle, after Crushing to $\frac{1}{8}$ In. Maximum Diameter.*

(Ash equals percentage of dry coal.)

(Whole sample)	2 lb.			
(½ sample)	1 lb. 11.37 per cent. ash*	1 lb.		
(¼ sample)	8 oz. 11.47 per cent. ash*	8 oz. 11.26 per cent. ash*	8 oz. 11.32 per cent. ash	8 oz.
(⅛ sample)	4 oz. 11.49 per cent. ash*	4 oz. 11.45 per cent. ash*	4 oz. 11.25 per cent. ash	4 oz. 11.26 per cent. ash
(1/16 sample)	2 oz. 11.31 per cent. ash*	2 oz. 11.66 per cent. ash*	2 oz. 11.43 per cent. ash	2 oz. 11.47 per cent. ash
(1/32 sample)	1 oz. 11.32 per cent. ash*	1 oz. 11.30 per cent. ash*	1 oz. 11.26 per cent. ash	1 oz. 12.06 per cent. ash
(1/64 sample)	½ oz. 11.07 per cent. ash*	½ oz. 11.56 per cent. ash*	½ oz. 11.43 per cent. ash	½ oz. 11.16 per cent. ash
(1/128 sample)	¼ oz. 11.43 per cent. ash	¼ oz. 10.70 per cent. ash	¼ oz. 11.91 per cent. ash	¼ oz. 11.20 per cent. ash

*Calculated from two halves of sample

standard, having a screw top about which tire-tape is wound, making them air-tight. S. W. Parr and W. F. Wheeler[2] have shown that coals deteriorate rapidly after mining and exposure to air. The nature of these losses is discussed on later pages by Messrs. Barker and Wheeler. Because of this deterioration it is desirable that samples be analyzed as quickly as possible after mining, and that they be kept air-tight. Coal-samples are usually kept in the Lightning or Mason jars. Parr and Wheeler have shown that the Mason jar does not make as perfect a seal as the Lightning jar.

In *Bulletin* No. 17 of the Engineering Experiment Station of the University of Illinois, S. W. Parr and N. D. Hamilton showed that coals submerged in water deteriorate but little, and while this method of keeping samples is not customary, it seems to be a very good one.

Methods of Analysis.

Air-Drying Loss.—The sample should weigh approximately 700 g. The air-drying loss is the loss in weight which the sample suffers upon drying at room-temperature for from 24 to 48 hr. It may be explained that air-drying loss is determined merely to bring the sample into equilibrium with the surrounding air as regards moisture, so that it can be weighed without subsequent loss or gain of moisture seriously affecting accuracy.

Oven-Drying Loss.—After having determined the air-drying loss, the sample is ground in a jaw-crusher to buckwheat-size, quartered to about 150 g., pulverized on a bucking-board so as to pass a 60-mesh sieve, and placed in a 0.5-pint Lightning jar. It is next thoroughly shaken, and 1 g. is weighed into a weighing-bottle of about 10 cc. capacity, the glass stopper of which fits closely over the top of the bottle. This bottle, with lid off, containing the coal, is heated at 105° to 108°C. for one hour, either in a toluene or an electric oven, after which the stopper is replaced and the bottle transferred to a desiccator, and allowed to cool, and then it is weighed. The loss in weight is called oven-drying loss.

Ash.—Either the residue from the determination of oven-drying loss or a fresh sample is used for this determination. In either event it is placed in a weighed porcelain crucible and

TABLE III.—*Analyses of Coal No. 5, in Saline County. (7 Samples.)*

	As Received.			Oven-Dry.		
	High.	Low.	Aver.	High.	Low.	Aver.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Moisture	6.64	4.43	5.90	....	....	....
Vol. matter	36.20	33.48	34.69	38.52	35.66	36.88
Fixed carbon	52.82	47.87	50.41	55.25	50.94	53.66
Ash	10.89	7.17	8.98	11.58	7.62	9.55
Sulphur	3.30	2.19	2.60	3.52	2.30	2.77
B.t.u.	12,883	12,159	12,552	13,700	12,942	13,197

TABLE IV.—*Analyses of Coal No. 6 from St. Clair, Madison, and Clinton Counties. (21 Samples).*

	As Received.			Oven-Dry.		
	High.	Low.	Aver.	High.	Low.	Aver.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Moisture	15.91	9.41	12.30	....	....	....
Vol. matter[a]	40.80	29.95	35.92	45.05	31.72	40.94
Fixed carbon[a]	45.50	37.43	40.68	52.75	42.91	46.46
Ash	14.26	9.33	10.84	16.56	9.69	11.72
Sulphur	4.59	1.39	3.55	5.29	1.65	4.01
B.t.u.	11,523	9,916	10,965	12,982	11,639	12,500

[a] Determined for only 18 samples.

Analyses of best and poorest samples, based on B.t.u. as received.

	As Received.		Oven Dry.	
	Best.	Poorest.	Best.	Poorest
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Moisture	9.44	14.81	....	....
Vol. matter	40.80	30.87	45.05	36.24
Fixed carbon	39.59	40.21	43.72	47.20
Ash	10.17	14.11	11.23	16.56
Sulphur	3.96	2.55	4.37	2.99
B.t.u.	11,523	9,916	12,723	11,639

heated slowly for half an hour over a low Bunsen flame. By this method all the volatile matter is driven off and the coal does not coke. Next, the flame is raised and the coal stirred with a platinum wire to hasten the combustion of the remaining carbon. After this, the crucible is put in the blast for half an hour, with occasional stirring to insure complete

TABLE V.—*Analyses and Heat-Values of Illinois Coals.*

County.	Local No. of Seam.	Mine Sample.											Laboratory Number.
		As Received.						Oven-Dry.					
		Moisture.	Volatile Matter.	Fixed Carbon.	Ash.	Sulphur.	B.t.u.	Volatile Matter.	Fixed Carbon.	Ash.	Sulphur.	B.t.u.	
		Per Ct	Per Ct	Per Ct	Per Ct	Per Ct		Per Ct	Per Ct	Per Ct	Per Ct		
Christian......	5	11.82	....	....	11.90	4.15	10,760	....	....	13.50	4.71	12,203	742
Fulton.........	5	15.09	35.39	38.09	10.63	3.21	10,573	41.68	45.80	12.52	3.79	12,450	1,404
Franklin......	7	10.62	....	....	7.87	0.67	11,751	....	....	8.82	0.76	13,148	419,420
Franklin......	7	14.40	....	....	6.90	1.02	11,459	....	....	8.08	1.19	13,400	461
Gallatin	5	4.47	36.10	49.07	10.36	3.56	12,645	37.82	51.33	10.85	3.72	13,235	1,092
Gallatin	5	4.30	....	....	10.74	4.36	12,452	....	....	....	....	....	361
Grundy........	2	14.69	....	....	6.97	3.09	11,276	....	....	8.17	3.62	13,217	733
Grundy........	2	11.16	....	....	5.00	1.57	11,531	....	....	5.82	1.83	13,436	734
Logan	5	14.80	....	....	11.76	3.03	10,586	....	....	13.81	3.56	12,426	720
Macon.........	5	13.91	37.00	39.33	9.76	3.29	10,804	42.95	45.72	11.33	3.82	12,549	1,569a
Macoupin ...	5	12.17	....	....	10.61	4.81	10,805	....	....	12.87	5.48	12,303	737
Macoupin ...	5	12.75	....	....	9.73	4.26	10,829	....	....	11.13	4.89	12,414	736,735
Macoupin ...	5	12.80	....	....	10.40	3.78	10,847	....	....	11.90	4.33	12,440	738
Peoria	5	14.73	35.92	36.74	12.61	3.38	10,451	42.13	43.09	14.78	3.97	12,257	1,410
Peoria	5	13.45	31.81	37.32	14.42	3.09	10,398	40.22	43.12	16.66	3.58	12,014	1,409
Perry..........	6	10.37	36.88	40.27	12.48	3.52	11,018	41.14	44.94	13.92	3.93	12,293	1,614
Perry..........	6	9.87	38.29	39.35	12.49	2.96	11,051	42.46	43.68	13.86	3.28	12,261	1,615
Perry..........	7	9.31	....	....	13.33	0.89	11,047	....	....	14.71	0.98	12,181	421
Perry..........	7	11.03	34.06	43.58	11.33	0.90	11,079	38.27	49.10	12.63	1.01	12,453	1,523
Perry..........	6	11.11	35.82	38.94	14.13	3.86	10,542	40.29	43.82	15.89	4.34	11,859	1,591
Perry..........	6	10.49	38.57	40.01	10.90	3.44	11,064	43.09	44.71	12.17	3.84	12,361	1,592
Randolph....	6	9.93	37.48	40.51	12.08	4.83	11,028	41.62	44.97	13.41	5.36	12,245	1,616
Randolph....	6	10.72	36.63	39.65	13.00	4.53	10,694	41.03	44.42	14.55	5.07	11,978	1,610
Saline[a]	5	4.89	36.53	47.65	10.93	3.96	12,298	38.41	50.10	11.49	4.16	12,931	1,095
Saline[a]	5	4.34	35.79	47.75	12.12	5.85	12,321	37.41	49.91	12.68	6.12	12,879	1,110
Saline...... }	7	5.73	....	....	11.19	3.13	12,177	....	....	....	....	....	360
and }	7	5.98	35.22	45.84	12.96	3.51	11,757	37.46	48.75	13.79	3.73	12,505	1,120
White...... }	7	6.71	36.75	45.81	10.73	4.16	11,889	39.39	49.11	11.50	4.46	12,744	1,121
Sangamon...	5	14.96	....	....	9.38	3.87	10,747	....	....	11.04	4.55	12,640	722
Sangamon...	5	13.71	....	....	10.82	3.62	10,691	....	....	12.61	4.19	12,392	741,740
Sangamon...	5	13.56	....	....	9.29	4.13	11,019	....	....	10.76	4.78	12,749	540
Sangamon...	5	14.39	....	....	11.68	3.95	10,534	....	....	13.64	4.61	12,301	721
Sangamon...	5	13.14	....	....	10.62	4.36	10,746	....	....	12.23	5.03	12,372	739
Tazewell.....	5	14.30	36.74	39.11	9.85	3.34	10,875	42.87	45.64	11.49	3.90	12,690	1,412
Tazewell.....	5	14.35	36.95	38.04	10.66	3.02	10,709	43.14	44.41	12.45	3.53	12,504	1,413
Vermilion....	6	12.96	37.26	42.79	6.99	1.55	11,580	42.80	49.17	8.03	1.78	13,304	558
Vermilion ...	6	12.56	36.92	42.52	8.00	1.23	11,418	42.25	48.60	9.15	1.41	13,058	557
White..........	...	See Saline.											
Williamson..	5	6.29	36.72	46.99	10.00	3.61	12,251	39.20	50.12	10.68	3.86	13,073	896
Williamson..	7	9.69	32.11	48.96	9.24	1.05	11,810	35.56	54.21	10.23	1.16	13,077	1,613
Williamson..	7	10.15	32.95	48.16	8.74	0.95	11,887	36.67	53.60	9.73	1.06	13,229	1,611
Williamson..	7	6.12	37.56	43.40	12.92	4.15	11,698	40.02	46.22	13.76	4.42	12,461	1,612
Williamson..	7	6.69	34.38	49.00	9.93	2.33	12,141	36.83	52.52	10.65	2.50	13,016	1,088
Williamson..	7	9.50	....	....	9.16	1.02	11,836	....	....	10.13	1.12	13,078	460
Williamson..	7	6.80	35.89	46.27	11.04	2.76	11,918	38.50	49.66	11.84	2.96	12,788	1,567
Williamson..	7	9.99	....	....	7.63	0.92	11,992	....	....	8.48	1.03	13,323	459
Williamson..	7	9.39	....	....	6.94	1.71	12,211	....	....	7.66	1.89	13,475	462

[a] At some distance from mines of Table III.

combustion, and then it is weighed and the unburned residue reported as ash.

Volatile Matter.—1 g. of coal is weighed into a tared platinum crucible, with evenly-fitting cover, placed on a platinum triangle and heated 7 min. by means of a Bunsen burner, having a flame 20 cm. high. The distance from the bottom of the crucible to the top of the burner should be about 7 cm. After weighing, the volatile material is calculated by subtracting the moisture from the loss in weight due to heating.

Fixed Carbon.—The fixed carbon is determined by calculation and is the result obtained by subtracting the moisture, ash, and volatile matter from 100.

Calorific Value.—Determinations of the heating-power in terms of B.t.u. are made with a Mahler calorimeter under carefully standardized conditions.

Analyses of Illinois Coals.

The State Geological Survey has determined the composition and heating-value of the Illinois coals in the seam for many localities by face-sampling and analysis, according to the uniform methods above described. The laboratory-work has been done by W. F. Wheeler and J. M. Lindgren under the direction of Prof. S. W. Parr. Most of the samples have come from scattered localities, and are only approximately representative of the seam for particular districts because of variations which occur, locally, from mine to mine. The available results of this general study are presented in Table V. More detailed studies have been made in connection with quadrangle surveys in the Saline county and Belleville regions. These show great uniformity in the first field, which involves Coal No. 5 (Table III.), and considerable variation in the latter field, which covers parts of St. Clair, Madison, and Clinton counties, and from which the Blue Band or so-called No. 6 coal is produced (Table IV.).

IV. The Occluded Gases in Illinois Coals.

By Perry Barker.*

In connection with the investigations on coal-deterioration conducted by the State Geological Survey and the Engineering Experiment Station of the University of Illinois, some determinations of the nature and amounts of occluded gases in Illinois coals have been made.

The first facts of importance in regard to these gases occluded or mechanically inclosed in the coals of Illinois developed when a number of fresh mine-samples were left tightly sealed for ten months. Upon opening at the end of this time, the gases in the containers ignited and burned with considerable flame.[1]

* Assistant Chemist, Engineering Experiment Station of the University of Illinois.

[1] *Trans.*, xxxviii., 630 (1908).

In order to study the composition of the gases thus evolved, several samples were collected from fresh seam-faces in a similar manner and allowed to stand for seven months. They were then opened under water, and the gas surrounding the coal in the containers was collected by displacement. A similar set of samples was collected in jars which were filled with water, and whatever gas had been given off at the end of seven months was collected. The results of these two sets of analyses are given in Table I.

TABLE I.—*Occluded Gases in Illinois Coals.*

	I.	II.	III.	IV.	V.	VI.
Weight of coal, grams,	642	800	648	800	787	800
Volume of gas, cc., [a]	446[b]	38.8	442[b]	29.3	331[b]	97.8
Per cent. by volume.						
CO_2	0	0	5.88	0	0.88	0
O	18.50	1.87	7.64	1.03	0	1.08
CH_4	0	55.97	0	35.39	11.83	90.28
N	81.50	42.16	86.48	63.58	87.29	8.64

I. Lebanon, Lebanon City Coal Co., sealed, dry.
II. Lebanon, Lebanon City Coal Co., sealed, submerged.
III. Bennett, Bennett Mine, International Coal Mining Co., sealed, dry.
IV. Bennett, Bennett Mine, International Coal Mining Co., sealed, submerged.
V. O'Fallon, Mine No. 2, St. Louis & O'Fallon Coal Co., sealed, dry.
VI. O'Fallon, Mine No. 2, St. Louis & O'Fallon Coal Co., sealed, submerged.

[a] At normal temperature and pressure.
[b] Total gas from containers.

It is evident that aside from the addition of methane and carbon dioxide to the ordinary constituents of air, a decrease in the percentage of oxygen originally contained in the air of the jars has taken place. In order to test the extent of this absorption of oxygen, a number of samples of coal were placed in jars with large volumes of air. These samples were portions of the series that had been sealed for ten months and later had been partly air-dried. Table II. shows the general nature of this change.

The two sets of analyses in Tables I. and II. give some indication of the nature of the alterations that are going on when coal is exposed, but give no information as to the composition of the gas remaining in the coal. Moreover, the samples were of various sizes of coal that had been broken

TABLE II.—*Occluded Gases in Illinois Coals.*

	I.	II.	III.	IV.	V.	VI.	VII.	VIII.
Weight of coal, grams,	109	139	180	183	146	134	138	153
Volume of gas, cc., [a]	873	849	816	814	843	853	850	837
Per cent. by volume.								
CO_2. . . .	0.48	0.94	0.68	1.87	0.25	1.23	1.11	1.62
O	0.16	0.13	0	0	0.25	0	0	1.45
CH_4	0	0	6.28	0	2.17	0	0	0
N	99.36	98.93	93.04	98.13	97.33	98.77	98.89	96.93

I. Springfield, Sangamon Mine, Sangamon Coal Co.
II. Springfield, Sangamon Mine, Sangamon Coal Co.
III. Eldorado, Mine No. 8, O'Gara Coal Co.
IV. Marion, Chicago & Big Muddy Coal Co.
V. Herrin, Squirrel Ridge Mine, Chicago & Carterville Coal Co.
VI. Duquoin, Greenwood, Davis Coal Co.
VII. Belleville, Suburban Coal Mining Co.
VIII. O'Fallon, Mine No. 2, St. Louis & O'Fallon Coal Co.

[a] At normal temperature and pressure. Total gas from containers.

from the face of the seam, and they had been exposed, even if only for short periods. In order to get coal closely representative of the material as it occurs in the seam, a set of samples of drill-dust were collected in the following manner:

As the drillings fell from the hole, they were collected in an ordinary 0.5-liter fractionating-flask fitted with a stop-cock at the side tube. When the flask was filled, it was sealed with a rubber stopper which was coated with a rubber-resin vacuum cement. These flasks were taken to the laboratory as soon as possible, and all the gases contained therein were removed by means of a mercury air-pump, and collected over mercury. The flasks were then allowed to stand for several days, after which they were again connected with the air-pump, and any gas that had been evolved was removed.

In order to have some extreme types of laboratory-weathered samples to compare with the fresh drillings, a set of coals that had been used for some previous tests were evacuated in the above manner. These were portions of mine-samples about two years old, and had been quartered, reduced to buckwheat-size, and air-dried. These two series correspond as to location of the mines, so that comparison of the changes in the occluded gases can be made by inspection of Table III.

The striking feature of the analyses in Table III. is the large

TABLE III.—*Occluded Gases in Illinois Coal.*

Last portions of air.

	I.	II.	III.	IV.	V.	VI.	VII.	VIII.	IX.	X.
Time of standing, days,	7	9	14	2	13	4	13	6	7	1
Weight of coal, grams,	261	209	220	205	244	204	217	204	231	108
Volume of gas, cc.,[a],	141.2	96.6	192.1	33.5	287.4	40.6	160.6	63.3	197.2	38.4
Cc. of gas per 100 g.,[b],	54.21	46.2	87.32	16.37	117.8	19.91	74.00	30.97	85.50	35.55
CO_2	2.12	1.91	3.37	1.27	6 55	1.23	3.27	0.54	10.34	0
O	2.87	2.06	0.94	2.54	0.58	2.75	0.95	5.04	0.95	7.90
CH_4	12.22	0	19.01	0	38.22	·0	1.57	0	19.81	0
N	37.00	42.23	64.05	12.56	72.45	15.93	68.21	25.39	54.40	27.76
Per cent. by volume.										
CO_2	3.92	4.15	3.86	7.80	5.56	6.20	4.43	1.79	12.09	0
O	5.30	4.46	1.04	15.50	0.49	13.80	1.28	16.25	1.11	22.20
CH_4	22.53	0	21.79	0	32.44	0	2.12	0	23.17	0
N	68.25	91.39	73.31	76.70	61.51	80.0	92.17	81 96	63.63	77.80

Gas removed by vacuum.

	I.	II.	III.	IV.	V.	VI.	VII.	VIII.	IX.	X.
Time of standing, days,	13	8	12	10	13	12	13	11	13	10
Weight of coal, grams,	261	209	220	205	244	204	217	204	231	108
Volume of gas, cc.,[a],	26.9	14.9	48.8	1.9	76.4	5.8	20.4	20.5	26.0	1.1
Cc. of gas per 100 g.,[b],	10.31	7.11	22.18	0.93	31.30	2.84	9.4	10.04	11.26	1.02
CO_2	1.84	5.74	1.68	....	4.63	0.69	3.18	2.01	3.51	0.56
O	0.50	0.10	0.14	....	0.29	0.15	0	0	0.82	0.10
CH_4	6.14	0	19.15	....	22.20	0	0.09	0	2.17	0
N	1.83	1.27	1.21	....	4.18	2.00	6.13	8.03	4.76	0.36
Per cent. by volume.										
CO_2	17.85	80.50	7.58	....	14.79	24.20	33.84	20.0	32.09	54.60
O	4.83	1.30	0.61	....	0.92	5.90	0	0	7.32	9.90
CH_4	59.59	0	86.37	....	70.93	0	1.00	0	19.26	0
N	17.73	18.20	5.44	....	13.36	69.90	65.16	80.0	41.33	35.50

I. Springfield, Sangamon Mine, Sangamon Coal Co., drillings.
II. Springfield, Sangamon Mine, Sangamon Coal Co., face-sample, 2 yr. old.
III. Herrin, Squirrel Ridge Mine, Chicago & Carterville Coal Co., drillings.
IV. Herrin, Squirrel Ridge Mine, Chicago & Carterville Coal Co., face-sample, 2 yr. old.
V. Clifford, Mine No. 8, Big Muddy Coal & Iron Co., drillings.
VI. Clifford, Mine No. 8, Big Muddy Coal & Iron Co., face-sample, 2 yr. old.
VII. Marion, Mine No. 3, Peabody Coal Co., drillings.
VIII. Marion, Mine No. 3, Peabody Coal Co., face-sample, 2 yr. old.
IX. Westville, Mine No. 44, Dering Coal Co., drillings.
X. Westville, Mine No. 44, Dering Coal Co., face-sample, 2 yr. old.

[a] At 0° C. and 760 mm. pressure. [b] Figured to coal as sampled.

loss of combustible gases by the fresh drillings. While this amounts to as much as 30 cc. per 100 g. in the fresh samples, no such gases were detected in the old lots. However, it must be understood that the relative amounts of gas in coal from these various mines cannot be critically judged from these analyses, as some of the working-faces had been within short distances of long-standing exposures. (A universal shut-down in the Illinois coal-mines during April and a part of May, 1908,

made it impossible to get samples representative of continuous workings.) Some idea of the rapidity of transpiration of occluded gases from exposed faces can be gathered from the following data.

As a drill-hole was driven, the dust from the first 2.5 ft. was collected in one flask, while that from the last 3 ft. was sealed in a separate container. As can be seen in Table IV., the sample farther from the exposed face contained more occluded gas, and had less changes produced in what did remain.

In addition to the loss of combustible gases, the drill-samples showed more extensive absorption than did the laboratory-weathered ones. From this it may be concluded, either that the oxygen has entered into some combination with the coal itself, or that a reaction has taken place, resulting in the formation of carbon dioxide. The presence of considerable amounts of carbon dioxide in the gases from the fresh samples seems to bear out the latter conclusion, although this gas does not completely replace the oxygen of the air. It may also be possible that the carbon dioxide formed and taking the place of the occluded gases is only given off at higher temperatures. That this is true to some extent is shown by the fact that 69 per cent. of the gases removed from one of these fresh samples at 100° C. consisted of carbon dioxide.

It is certainly true that this absorption of oxygen takes place as soon as the gases escape from the fresh coal. A study of some of the stages of this absorption or oxidation can be made from Table V. In this table all samples were from the same mine. Nos. I. and II. were partly air-dried face-samples about two years old. From No. I. the surrounding air in the container was collected by displacement and analyzed. No. II. was left in one of the sealed fractionating-flasks for two days. At the end of that time both the surrounding air and some of the inclosed gases were removed by means of the air-pump. No. III. is a flask of drillings from which the surrounding air and occluded gases were removed as above. No. IV. is the analysis of the gas given off after the surrounding air had been removed and the flask had stood in a vacuum for 12 days. No. V. is the analysis of the air that had been admitted to the evacuated flask and left in contact with the coal for seven days.

The preceding results give some light upon the changes pro-

TABLE IV.—*Occluded Gases in Illinois Coal.*

	I.	II.	III.	IV.
Time of standing, days,	7	7	13	13
Weight of coal, grams,	182	231	182	231
Volume of gas at 0°C., 760 mm.,	174	197.2	9.6	26.0
Cc. of gas per 100 g.,	95.30	85.47	5.27	11.26
CO_2	7.21	10.34	1.76	3.51
O	0.59	0.95	0	0.82
CH_4	11.28	19.81	2.30	2.17
N	76.22	54.37	1.21	4.76
Per cent. by volume.				
CO_2	7.57	12.09	33.33	32.09
O	0.62	1.11	0	7.32
CH_4	11.73	23.17	43.74	19.26
N	80.08	63.63	22.93	41.33

I. Westville, drillings from first 2.5 ft. of hole, last air.
II. Westville, drillings from last 3 ft. of hole, last air.
III. Westville, drillings from first 2.5 ft. of hole, gas by vacuum.
IV. Westville, drillings from last 3 ft. of hole, gas by vacuum.

TABLE V.—*Occluded Gases in Illinois Coal.*

	I.	II.	III.	IV.	V.
Weight of coal, grams,	146	20.5	220	220	220
Volume of gas, cc.,	843	33.5	192.1	48.8	130.4
Per cent. by volume.					
CO_2	0.25	7.80	3.86	7.58	1.63
O	0.25	15.50	1.04	0.61	0.37
CH_4	2.17	0	21.79	86.37	14.14
N	97.33	76.70	73.31	5.44	83.86

I. Old face-sample in contact with large volume of air.
II. Old face-sample sealed 2 days.
III. Drillings, sealed 14 days.
IV. Drillings, in vacuum 12 days.
V. Drillings, second air in contact with coal 7 days.

duced by the deterioration of sealed laboratory-samples, but contain no data as to samples subjected to outside exposure. Table VI. gives a comparison between samples of fresh drillings and samples exposed to the weather. No. I. is a sample of drillings from Westville, while No. II. was collected off the surface of a pile of the same screenings 15 months old, and No. III. is a sample of the same screenings that had been stored outside for two months. No. IV. is a sample of drillings from Marion, while No. V. is from an outcrop of the same seam one mile from the place where No. IV. was taken. This outcrop

TABLE VI.—*Occluded Gases in Illinois Coal.*

Last portions of air.

	I.	II.	III.	IV.	V.	VI.	VII.
Time of standing, days,	7	3	6	13	19	7	3
Weight of coal, grams,	231	200	200	217	224	261	200
Vol. of gas, 0°C., 760 mm.,	197.2	44.2	55.8	160.6	134.3	141.2	240.5
Cc. of gas per 100 g.,	85.50	22.10	27.9	74.01	60.0	54.21	120.25
CO_2	10.34	1.40	0.25	3.27	3.38	2.12	3.03
O	0.95	3.25	5.75	0.95	0.38	2.87	22.09
CH_4	19.81	0	0	1.57	0.54	12.22	0.66
N	54.40	17.45	21.90	68.22	55.70	37.00	94.47
Per cent. by volume.							
CO_2	12.09	6.34	0.90	4.43	5.64	3.92	2.51
O	1.11	14.71	20.61	1.28	0.64	5.30	18.36
CH_4	23.17	0	0	2.12	0.54	22.53	0.55
N	63.63	78.95	78.49	92.17	93.18	68.25	78.58

Gas removed by vacuum.

	I.	II.	III.	IV.	V.	VI.	VII.
Time of standing, days,	13	14	13	13	7	13	13
Weight of coal, grams,	231	200	200	217	217	261	200
Volume of gas, 0° C., 760 mm.,	26.0	26.9	16.8	20.4	19.3	26.9	23.9
Cc. of gas per 100 g.,	11.26	13.45	8.4	9.40	8.88	10.31	11.95
CO_2	3.51	1.4	2.85	3.18	2.99	1.84	2.39
O	0.82	2.8	0.7	0	0.36	0.50	0.81
CH_4	2.17	0.15	0.6	0.09	0.10	6.14	2.39
N	4.76	9.10	4.25	6.13	5.43	1.83	6.36
Per cent. by volume.							
CO_2	32.09	10.41	33.93	33.84	33.67	17.85	20.00
O	7.32	20.82	8.33	0	4.02	4.83	6.78
CH_4	19.26	1.12	7.15	1.00	1.05	59.59	20.00
N	41.33	67.65	50.59	65.16	61.26	17.73	53.22

I. Westville, Mine No. 44, Dering Coal Co., drillings.
II. Westville, Mine No. 44, Dering Coal Co., screenings, 15 months old.
III. Westville, Mine No. 44, Dering Coal Co., screenings, 2 months old.
IV. Marion, Mine No. 3, Peabody Coal Co., drillings.
V. Marion, Binkley, Miles Co., outcrop coal.
VI. Springfield, Sangamon Mine, Sangamon Coal Co., drillings.
VII. Springfield, Sangamon mine, Sangamon Coal Co., screenings, 2 months old.

had been exposed for one year. No. VI. is a sample of drillings from Springfield, while No. VII. was collected from the surface of a pile of 1.5-in. screenings from the same mine. These screenings had been stored outside for two months.

Conclusions.

The conclusions drawn from these investigations may be summarized as follows:

1. Loss of combustible gas begins as soon as pressure upon the coal in the seam is released and air is brought into contact with the newly-exposed surfaces.

2. As soon as the gases occluded by the coal are released, an

absorption of oxygen from the atmosphere begins. The oxygen may enter the coal substance and combine with it or may unite with carbon to form carbon dioxide.

3. Carbon dioxide is undoubtedly formed to some extent by the action between the coal and whatever oxygen it had absorbed.

4. Upon outside exposure, coal loses most of its occluded gases and even a large part of the carbon dioxide formed by the absorption of oxygen.

Experiments at higher temperatures will be conducted in order to determine, if possible, more exactly the changes that are produced when coal deteriorates, and to throw some light on the cause of spontaneous combustion.

V. Mining-Wastes and Mining-Costs in Illinois.

By George S. Rice.*

Introduction.

In coal-mining operations throughout the State of Illinois there is a greater range in the amount of coal extracted from a given volume of coal-seam than might be expected from the remarkable uniformity in thickness of the chief seams. The percentage of yield varies from about 50 to more than 95 per cent. The latter high yield is obtained in the long-wall mines of northern Illinois, embracing the Wilmington and the so-called "Third Vein" fields. The coal-seam mined in these two districts is the same geologically—the No. 2 of the Worthen Survey. The lowest yield is from the thick, more deeply buried, Blue Band seam of central and south central Illinois, in some localities termed No. 5, in others No. 6.

The variation in yield from the total amount of coal under a given land-surface is still greater, for in the thick-seam districts there are usually other seams than the one worked that it is possible, physically, to work. These are, in many cases, rendered more or less unworkable, when the distance between the seams is small, by being undermined.

Causes of Mining-Waste.

General.—The influencing conditions causing the great losses that are at present incurred are:

* Consulting Engineer, Chicago.

1. Cheapness of "coal in place;" that is, in the seam.
2. Low market-prices, resulting from extreme competition.
3. Character of the seam, roof, and floor as determining the method of mining.
4. Surface-subsidence due to mining.
5. Interlaced boundary ownerships.
6. Carelessness in mining-operations.

The first two factors, taken together, are the controlling ones in most mining-operations in influencing the choice of a mining-system. The majority of Illinois operators are sufficiently progressive to find ways and means to take out practically all the coal under a given area if it could be made evident that it paid to do so. That many do not do all that can be done in this direction is apparent; but if, without unusual investment, a profit of operation could be shown in taking out all the coal over the profit made by present methods, the industry could undoubtedly find men to accomplish the task. In other words, from an engineering standpoint practically all the coal under a given area can be taken out. It is a question of cost.

Cheapness of Coal in Place.—This is chiefly due to the great abundance of coal. Except in the barren northern one-fourth of the State, lying north of the outcrop of the coal-basin, the development of a tract depends primarily not on the possibility of finding coal in that particular locality, but on the question whether it is a suitable place, from a market stand-point, to open a mine, the thickness of seam and the quality of the coal being considered.

The price of coal-rights varies from $10 per superficial acre in the middle part of Illinois, away from the mining-centers, to $100 per acre near developed mines. Or, in the case of leasing, from 2 cents per ton run-of-mine hoisted, in the southern part of the State, to 5 cents in the northern part. The cost of the fee is relatively so much cheaper per ton than leasing that the latter system is not much used. The ownership of the coal by the operator is conducive to better mining, but relative to other items that go to make up the total, the cost of the "coal in place" is so low as to be almost negligible. In central Illinois, in some cases, at a cost of only $10 per acre, two workable seams, from 6 to 8 ft. thick, are obtained. Allowing only 50 per cent. yield of the two seams, 13,000 tons would be produced

per acre, the purchase cost thus being $\frac{1}{13}$ of a cent per ton, or about $\frac{1}{1000}$ of the total cost of production in central Illinois. In the Wilmington long-wall field the average cost of the coal-rights is about $50 per acre. The seam there, although it averages a trifle less than 3 ft. in thickness, produces about 5,000 tons per acre. The cost is therefore about 1 cent per ton in place, which is $\frac{1}{130}$ of the total cost of production. Hence, it may be seen that there is little incentive, from the standpoint of the purchase-price of the coal, to save the latter in mining-operations.

Low Market-Prices.—The tremendous development of the coal-carrying railroads and the policy of making low ton-mile rates for long hauls has resulted in excessive competition, both from within and from without the State. The cheaply-produced coals of the Eastern States, and particularly West Virginia, resulting from favorable natural conditions and lower labor-cost, with low through freight-rates, have enabled them to enter the natural coal-markets of Illinois and sell at prices very little above what the Illinois coals bring. The high quality of these coals, particularly those that make little smoke, has allowed them to set the pace in making prices.

The competition between the Illinois coals has been even more severe. This results from the multiplicity of ownerships, due mainly to the ease of opening new mines. Each period of unusual prosperity in the Western coal business, like that at the time of the anthracite strike, is followed by an immense increase in capacity. For example: In 1906 and 1907 railroad shipping-mines operated an average of 190 and 195 days, during the respective years, out of 300 working-days; in other words, only 63 and 65 per cent. of the time (see Figs. 1 and 2). To a certain extent this is unavoidable, as the markets are in a climate of extreme cold in winter, and as the Illinois coal stocks very indifferently, the winter demand tends to fix the capacity. This, in turn, makes the labor-rates high, to cover the period of idleness. On the other hand, it makes severe competition during the spring and summer months, in the effort of each operator to keep his mine running as much as possible.

The annual coal-report of Illinois for 1906, compiled by the Bureau of Labor Statistics, gives as the average value, or selling-price, per ton, of all sizes on cars at the mines, for northern

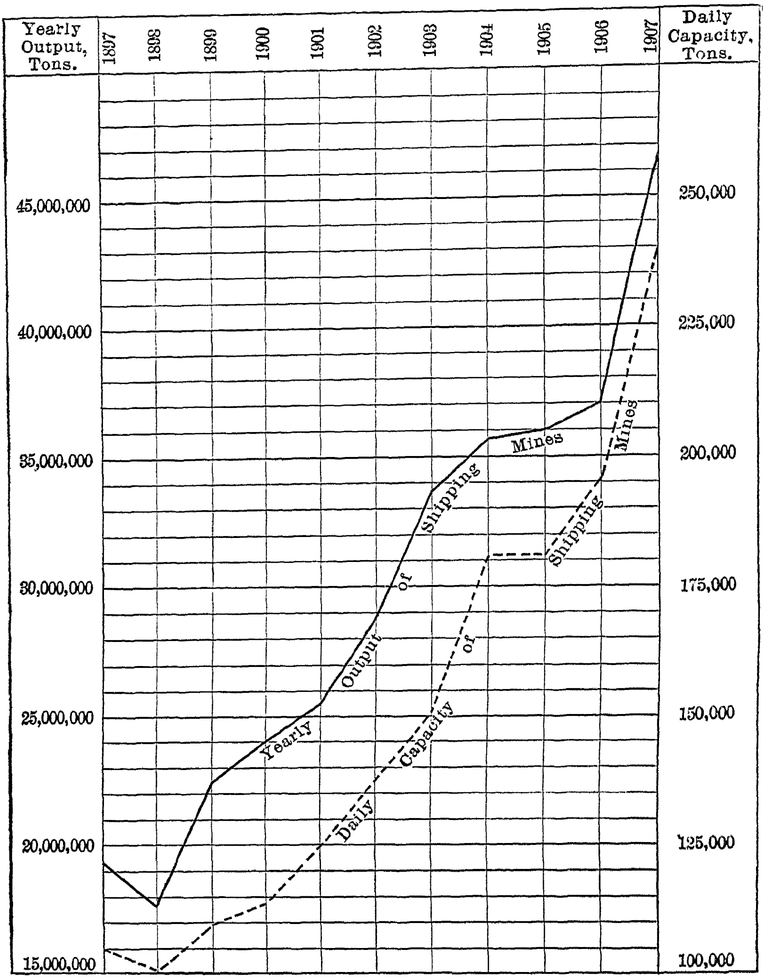


FIG. 1.—ILLINOIS SHIPPING-MINES: YEARLY OUTPUT OF COAL, ALSO DAILY CAPACITY FOR A PERIOD OF 10 FISCAL YEARS.

long-wall districts, $1.41; for the "shooting-coal" districts, from $0.866 to $1.153; and for the whole State average, $1.029. As the State treats the individual mine-returns as confidential, the figures given are generally regarded by operators as essentially correct. The average hand-mining rates for the long-wall districts are $0.754 and $0.784, and of the "shooting-coal" districts from $0.458 to $0.609. The underground hauling, timbering, brushing, hoisting, top-labor, and supplies must be added to the foregoing figures to obtain total

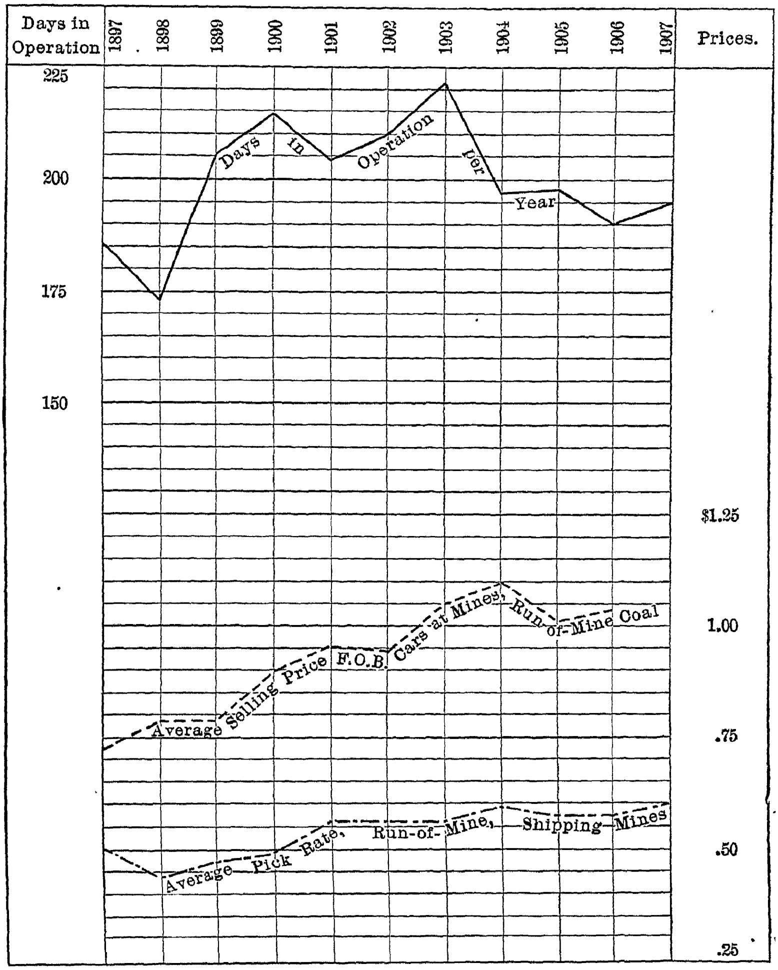


FIG. 2.—ILLINOIS SHIPPING-MINES: DAYS IN OPERATION, AVERAGE VALUE, AND PICK-RATE, YEARLY, FOR A PERIOD OF 10 FISCAL YEARS.

operating-costs. The average total cost per ton of coal loaded on cars, including general and selling expense and amortization, but not capitalization interest-charges, is from $1.20 to $1.30 for the coal produced in the long-wall districts, and from $0.70 to $0.95 for the other districts. It is safe to say that the average net profit per ton throughout the State for a whole year does not exceed 20 cents, and if the interest on the capital be taken out, the average profit will be reduced to 10 or 15 cents per ton; the actual profit is probably less than these figures.

It will therefore be seen that, with the small margin of profit, there is little incentive for the individual mine-operator of Illinois to conserve, beyond customary good practice, the coal he owns or leases.

Character of Seam, Roof, and Floor, as Determining the Method of Mining.—It is the system of mining adopted that determines the proportion of coal won in a given seam. Where the long-wall system can be physically and commercially used, the problem is solved, for in a well-handled long-wall mine the only loss of coal is in the fine particles, which become mixed with the fire-clay and roof-dribblings and get shoveled back into the gob. Probably not more than 2 per cent. is lost in this way.

Under the ordinary long-wall conditions of northern Illinois, falls of roof, especially during periods of idleness, local carelessness in leaving "points" or projections of the face, and abandonment of corners in the ownership of the land cause additional losses. In one mine, at which a record was kept for a period of six years, the total loss of coal from all causes was 5 per cent.

About 5,300,000 tons were produced in 1906 by long-wall mines, nearly all in the Wilmington and the Third Vein districts, at the north end of the Illinois field. The output of these districts has been practically stationary for some years, owing to the competition of Eastern coals and of the thick-seam coals of middle and southern Illinois. Long-wall is the only system that can be successfully used in the No. 2 seam, as found in the northern districts. Briefly, the conditions are these:

A blocky coal when mined by undercutting, but tender and flying to pieces when "shot" down; a "soapstone" (shale) roof without fissure-cracks, until such are formed by the successive settlements caused by the undercutting; a clay under the coal that generally presents fairly-easy cutting; and a harder sandy clay floor which causes the coal and under-clay to break or work, when the roof "weight" is properly thrown on the long-wall face, by systematic building of pack-walls and keeping the faces aligned. Finally, the mines, with one or two exceptions, are dry.

In the larger part of the Illinois field the "advancing long-wall" of the northern "thin-vein" field is not commercially practicable. The roof is generally too hard to "break" prop-

erly, and usually there is no "draw-slate" with which to make buildings. The other conditions, clay-mining and dryness, are all right, but the former are obstacles to "advancing" long-wall, without the extraordinary expense of importing stone for pack-walls.

The seams are considered below in their stratigraphic order.

No. 1 of the Worthen Survey is not generally identified throughout the State. Coal worked in the vicinity of Rock Island is called No. 1. It occupies channels and local depressions cut into the shale previously laid down on the Burlington limestone. It seems likely that this coal may belong to a later period than the shaly seam which quite regularly underlies the No. 2 seam in the Wilmington and Third Vein districts. The system of mining the former is the ordinary room-and-pillar method, the pillars in some cases being withdrawn. The yield is from 65 to 70 per cent. of the territory covered by the entries. The channels are usually narrow and the coal thins along the margin, so that all coal less than about 3 ft. thick is lost. As a whole, No. 1 seam has little commercial importance in the State.

No. 2 is a remarkably persistent seam, apparently extending throughout the whole of the Illinois basin. It varies in thickness from 1.5 to 4 ft. While a high-grade Illinois coal, the cost of production makes it commercially available only in the northern field, where it is extensively opened, as already described, by long-wall mines. Elsewhere there are large areas of this seam, running from 2 to 2.5 ft. in thickness. While this is too thin to work at present, it is not underlain by valuable coal, and hence will not be damaged by any mining-operations below it, but will remain as a reserve and a problem for future operators.

Nos. 3 and 4 seams are not well-defined horizons and are practically negligible, so far as at present known.

No. 5 is an important seam. It has been extensively developed in Fulton county and in the Springfield district, where it shows great uniformity. In the central and southern part of the coal-basin it is not clearly defined. Its characteristic feature is the presence of clay-slips running irregularly through the coal, and indicating shattering, with subsequent filling. It has a strong slate roof, which is more or less sandy, and pre-

sents a pebbly, knobby surface when exposed in the roof; usually there is a "draw-slate" between roof and coal, which is sometimes strong enough to be held up by timbering. The coal itself is clean. The irregular clay-slips, "horses," and sulphur-balls are frequent but are separable. The floor is a shale, which exfoliates when exposed to the air.

In the Fulton county field this seam is from 4 to 5 ft. thick and remarkably even. It is mined by the room-and-pillar system. As it is shallow, the pillars are left very small, and, in general, are not pulled. A great deal of coal in the vicinity of the outcrops is rendered unworkable by nearness to valleys filled with glacial drift. Within the workable areas the yield by the present methods is from 60 to 65 per cent. of the coal in place.

In the Springfield district, No. 5 coal is from 5 to 6 ft. thick. The room-and-pillar system is employed. Some pillars are drawn, but generally the clay-slips and "horses" are so frequent that an effort is made to lay out the pillars to include them. On the whole, the yield is about the same as in the Fulton county field. South of Springfield, and in the Third Vein district, No. 5 coal is more pockety. As it is from 160 to 190 ft. above seam No. 2, and as the latter is worked only long-wall, the unworked areas will not be damaged by working out the lower seams.

No. 6 is the great producing seam of the State. Except for a few developments in northern Illinois, in Bureau county, the mines working it are in central and southern Illinois. The seam lies from 40 to 60 ft. above the No. 5 horizon. It is characterized by a "blue band," which occurs from 2 to 4 ft. above the bottom. The seam is from 5 to 9 ft. thick. The Virden-Mt. Olive seam and the Duquoin-Ziegler seam both belong, in my opinion, to the Blue Band seam. The seam in the former district has generally been called No. 5, although drilling indicates a pockety seam below it at the No. 5 horizon. The State Geological Survey's recent investigations, I am informed, show that the Herrin seam in Williamson county, heretofore called No. 7, also belongs to the No. 6 horizon.

The main roof of the Blue Band seam is a limestone, usually with a shale or clay layer, 1 ft. to 4 ft. thick, between it and the coal. In some places the limestone comes down to the coal, in others it disappears entirely. In all cases the main

roof is very strong, and this has an important bearing on the system of mining adopted. The coal is usually a brighter, cleaner coal in itself than the No. 5 coal, but in central Illinois it has a large amount of "sulphur" (iron pyrites) in balls, lenses, and stringers. The coal, and sometimes the roof of this seam, contains considerable marsh gas in the more deeply buried portions. The floor is a clay containing, below the mining, many "nigger-heads." This clay readily "squeezes" when subjected to pressure, as when pillars are too small. The main roof is very difficult to "break," so that, when the coal is from 350 to 600 ft. or more deep, it is very essential to leave large pillars. Many mines, in other ways well systematized, have had serious difficulty from "squeezes" brought on by leaving too-thin pillars or robbing them. The older mines were all opened on the ordinary room-and-pillar plan. Lately, the majority of the mines have been changing to the "panel" system with beneficial results, but as yet no systematic attempts have been made to pull pillars. The result is, that in the deeper mines only one-half of the coal is secured, taking into account barrier as well as other pillars.

As a whole, there is taken out of this seam only from 50 to 60 per cent. of the coal it contains, the gross quantity of which throughout the State certainly exceeds the contents of any other seam, and possibly of seams Nos. 5 and 7 together. It is the seam now most actively worked, and will continue to be the most productive; but its exploitation involves the largest losses, and hence calls for the most earnest study in the interest of greater economy. The coal, when freed from impurity, is a very strong steam-coal, and the seam tends to improve in quality as well as thickness in going south.

No. 7 is not generally present in northern Illinois. If it ever was so, it has been generally eroded. Outside of the Danville district, it certainly does not exist as a thick seam. In that district it is known as the Danville seam. The Grape Creek seam, which is below it, and in which there has been the greatest development, is considered to be No. 6. The correlation of these various seams by the State Survey is looked forward to with great interest. As mined at Danville, No. 7 is from 5 to 7 ft. thick. It has been opened up by the usual room-and-pillar system, and the pillars are not drawn. Owing

to the numerous slips and poor roof the yield is low, probably but little over 50 per cent.

There have been no workable seams discovered above the No. 7. Here and there in the upper part of the deep sections of the measures in the central part of the basin, as found by drilling, there are seams from 1 to 2 ft. thick, but the time is very remote when they can be made available.

The Surface Subsidence Due to Mining.—The influence of this factor upon the yield results from the high value of Illinois lands for agricultural purposes, well-improved, tiled land being worth from $100 to $175 per acre. In mining coal by the long-wall system, where the overlying surface is flat, the tile-drains are deranged and swampy places are made, although the surface may sink only from 1.5 to 2 ft. This is particularly the case in the Wilmington district. In the Lasalle district the ground is more rolling and the subsidence has little effect. Although the goaves in the mines of these districts are supposed to be nearly filled, in reality they are not, and the settlement is practically one-half the thickness of the seam. If the long-wall system were applied to the thick seams, when applicable at all, it would cause a considerable derangement of the surface, and where the latter is so nearly level as the prairie-land of central Illinois, it makes the question of subsidence a serious one. It may be solved to a certain extent through draining the sunken lands by pumping, but even with such a method, aside from the expense, there is a serious difficulty from storm-water. When the subsidence of the surface is from 2 to 4 ft. it will render previously-level lands of little use for raising crops until the particular area has come to full settlement and has been re-tiled. The same is true if all the coal be taken out by any other system, and is even more emphasized where no pack-walls are used, because then the subsidence is practically the full thickness of the seam. If it were possible to systematize mining so that the land nearest the water-courses was first undermined, and then in succession the land further away, the damage done to farming would be minimized. However, until the agricultural land in the United States becomes insufficient to fill the needs of the population, which would be reflected in a continual increase of price for farming-land, the money-loss from temporarily destroying

the surface in places is relatively small, as compared with the selling-price of the coal mined from the seam. Taking the average value of the surface at $125 per acre, if 80 per cent. be rendered worthless, the immediate money-loss would be $100 per acre. A seam 6 ft. thick would contain per acre 11,000 tons of coal in place, yielding, at 90 per cent., 9,900 tons. The damage done by practically destroying the surface would be only 1 cent per ton. If the land-prices should rise two or three times above the value stated, this loss would still not prohibit mining.

Interlaced Boundary Ownership.—The losses in mining arising from this factor have been perhaps less important in the past than they may be in the future. I had occasion some time ago to examine a property where there was coal within a quarter of a mile of the shaft on either side, yet the operator was obliged to mine coal at a distance of one and a half miles. Some of that near the shaft could not be purchased, yet it lay in such a way near a geological uplift that it would not pay to open another mine to reach it. A number of such instances can be found. There have been large purchases, by various corporations, of coal-rights through Illinois, which have been taken up checker-work fashion, and unless the State interposes its authority, it is conceivable in some cases that coal may be lost through improper development due to adverse ownership. Such effects, in a number of cases, have been avoided by adjoining owners getting together and trading to form areas suitable for the lay-out of individual mines.

Carelessness in Mining-Operations.—Losses from this source have been very great. They may be set down under these heads:

1. Improper system of mining.

2. Carelessness in following any system of mining by which blocks of coal are lost, and, where pillars are being pulled, carelessness in failing to systematize the work, so that "squeezes" will be avoided. The same is true of advancing work, where improper proportioning of pillars or alignment of roads has brought on a "squeeze."

3. Inadequate surveys, records, and maps, so that, with change of underground management, there has been a failure to give proper notice of pillars and blocks of coal that temporarily have been passed by.

4. An entirely inadequate system for filing maps and survey-records of abandoned mines with either the county or the State authorities. The absence of definite knowledge compels a new adjacent mine, as a matter of safety, to keep farther away from an old abandoned mine, which may be full of water or gas, than would be necessary. At present it is impossible to find any map of a mine abandoned some years back. There is also insufficient attention paid to compelling operators of mines that are about to be abandoned to bring up the mine-surveys in a careful manner. In my opinion, the preservation of mine-maps is properly a function for the State, as it is now for a county to record deeds, and there should be a permanent bureau established for the proper recording of the surveys and maps of abandoned mines. This bureau also should take charge of and systematically file the maps of "going" mines.

Remedies for Waste in Mining.

Possible Saving.—I believe it is possible to take out from 90 to 95 per cent. of the coal under a given area, even if the character of the roof is such that pillars cannot be pulled and the advancing long-wall system is not applicable; that, in general, this can be done at an additional cost which need not make it prohibitive. In fact, taken over the whole life-time of the mine, it may be a profitable operation.

When advancing long-wall can be used, it is plainly the most direct system to apply, but, as already observed, it is inapplicable in Illinois, in most cases, outside of the No. 2 seam.

Filling System.—Two general systems suggest themselves, one of which is a replacement with material sent down from the surface. This method is more or less employed in the anthracite-district of Pennsylvania, where the culm-bank is used for the filling. It is also used extensively in Silesia. In Illinois, the substitute would have to be surface-sands and gravel. That this would be impracticable in the great majority of cases throughout the State is self-evident, particularly if water, the usual vehicle for transportation, is employed, inasmuch as the majority of the thick seams in Illinois have clay under them which water would soften and thus tend to cause a "squeeze." Aside from this, much farm-land would be destroyed in getting the filling material.

Retreating Long-Wall.—The other system is driving to the boundaries of the property and then using either retreating long-wall or semi-long-wall systems, such as have been extensively developed in England. One or the other of these systems, in my opinion, could be applied in almost all cases, meeting the obstacles of strong roof and clay floor. The difficulties of a retreating system are these: the delay in getting an output, the increase in capitalization, and the added cost in the early stages of the mine, due to the increased capitalization. The off-set would be the saving of the coal; but this is a minor item of expense, and is balanced by the damage to the surface.

Estimates of Costs.—Taking a theoretical case, the figures would be about as follows: Let us assume, in the Blue Band seam of central Illinois, coal averaging 7 ft. in thickness and 400 ft. deep; shafts in the center of a group of four sections of land (2,560 acres); a mine-equipment costing $125,000; town-site, coal-rights and miscellaneous outlays, as much more; making a total capitalization of $250,000. Assume a pair of entries to be driven to the middle of one side of the property, thence to a corner, with additional stubs, making a total of about 8,000 yards of single entry. The use of machinery for driving the entries is presupposed, both for the sake of speed and for the advantage of the undercutting, which allows the coal to be blasted down with small charges. Hence, three shifts could be used. Assuming an average advance of 5 yd. per day, the cost would be roughly about $10 per yard in excess of the value of the coal produced. This would make a total charge of $80,000 in excess of the ordinary cost of development. At a speed of 5 yd. per day it would take about 800 working-days to drive the pair of entries the 2 miles to the corner, plus some stubs. Allowing for the inevitable delays, this would mean 2.5 years to get to the same point of development ordinarily reached when the main- and-escape shafts have reached coal and have been connected underground. Allowing 6 per cent. per annum for half the period (1.25 years), the interest on $80,000 would be $6,000. Possibly the expenditure of $50,000 of the entire plant and town-site investment previously mentioned could be deferred till the final period of development. If so $200,000 would be drawing interest for 2 years and 6 months—say at 6 per cent. per annum. We then have:

Excess cost of 8,000-yd. entry complete, ...	$80,000
6 per cent. interest on $80,000 for 1 year and 3 months, .	6,000
6 per cent. interest on $200,000 for 2 years and 6 months,	30,000
Total excess cost of special development over that of ordinary development, ...	$116,000

Hence, this additional amount of capital would be required. At 6 per cent. the interest on $116,000 will constitute an annual charge of $6,960. If the theoretical plant has an annual average output of 300,000 tons, which would be normal for a commercial mine for the investment mentioned ($250,000), this fixed annual charge would amount to 2.32 cents per ton hoisted.

The cost of mining at the face, under the present labor-contracts, would probably be the same as it is under the prevailing system. The cost of hoisting, dumping, and loading also would be the same, but at the start certain other items would be greater; the "care-of-mine" cost, due to the keeping up of the first long roads, would be larger than at any subsequent period. The same would be true of haulage; it has been assumed that the cost of an electric haulage for the 2 miles (4 miles round trip) over what would be merely "gathering" in the conventional starting of a mine, at the center of the property, and which would be similar to the development at the corner of the property, should be between 4 and 5 cents per ton, considering labor, fuel, repairs, and sinking-fund for the haulage-plant.

During the life-time of the mine this cost would be constantly decreasing, instead of increasing, as in the conventional advancing mine. The average over the whole period would be practically the same. In the "care-of-mine" cost, however, the average during the whole life-time of the mine should be considerably less than that of the conventional advancing mine. How much less is conjectural, but that the saving would more than compensate for fixed charges arising from the greater first-cost of the retreating mine, I have no doubt. At the start, the situation would have to be faced that the cost of the coal would be still further increased something like this:

	Cents Per Ton.
Interest on additional capital, ...	2.32
Additional cost of haulage, ...	4.50
Cost of maintaining and ventilating four miles of single entry, labor, timber and fuel, $40 per day, ...	2.60
Total additional cost at start, ...	9.42

Effect of Introducing New System.—Under present market-conditions it would practically wipe out all profits for the average Illinois mine. This, together with the deferment of the time of getting the first returns, which ordinarily is from 1 to 1.5 years, to a total of 3.5 or 4 years, brings about a condition making it virtually impossible to enlist new capital to open a mine in this way. The plan seems feasible only for the largest companies, and these would gradually change; that is, start new mines on the retreating plan while operating their old properties on conventional lines. Evidently, large consolidations could best effect this purpose. To force an immediate change of old as well as new mines by State or national laws would be too drastic. To make the requirement for all new mines, if legally it can be done, would undoubtedly have the effect of restricting new developments. If the law were national in scope it would probably result in such curtailment of new output that very soon there would be a shortage of fuel and an increase in price until capital was again attracted. The effect would be even more severe in the mountainous States than in Illinois, inasmuch as the development of satisfactory methods to meet the physical disadvantages of steeply-pitching seams, or even level ones running under high mountains, will be difficult, and still more so where the seams are faulted.

Legislation Needed.—Much, however, can be accomplished by voluntary means and by the making of such laws by the State as would require the filing of proposed plans of development for any new mine, and their approval by a board of examiners before mining is allowed to begin, plans leading to unsafe conditions and too wasteful in method not being permitted. This would amount to the giving of free engineering advice by specialists; but if it aided in conserving the mineral resources of the State and the country it would be worth more than the relatively small cost of maintaining a bureau, either by the State, or jointly by the State and the national government.

Education Needed.—Much can be done by a campaign of education. This country has highly developed its coal-mining machinery, and in this respect has been in the front rank, enabling it to produce cheap fuel with relatively high labor-cost, but in the manner of laying out our mines underground and in directing the work at the face we have been practically stationary

for years. When unusual conditions are encountered in a mine, that part of the mine is too often abandoned. We have fires and disasters sometimes due to lack of knowledge, care, or thoroughness on the part of the underground foremen, who are usually striving to make a record for tonnage.

Having so much easily-mined coal, we have tended to avoid all adverse conditions, picking out the good spots. This has not developed our skill in meeting difficult conditions, so that we are undoubtedly far behind England and Europe generally in our work at the "face" of the mine. Our best mine-foremen have been trained abroad in a practical way, even if their schooling has sometimes been acquired here.

We have much to learn; and now that the government has started on its campaign of education it is to be hoped that Illinois and other mining States will awake to the call for "the conservation of natural resources."

VI. The Use of Illinois Coal for Domestic Purposes.

By J. M. Snodgrass.*

Under "domestic purposes," it is intended to include all heating, cooking, etc., by the burning of coal in stoves, ranges, and furnaces, or under house-heating boilers, in dwellings or other buildings. The amount of coal or other fuel so burned is large, and the question of its efficient combustion without undue smoke and dirt, and without troublesome fire-conditions, is directly important to a large number of consumers and to the community as a whole. A satisfactory solution of the problems connected with burning Illinois coal for domestic purposes would mean a very considerable saving for the consumer and a much better market both within the State and throughout the territory naturally supplied with this coal.

Until recently, this subject has received little consideration as compared with the use of coal for power and for heating upon a large scale. The manufacturers of stoves, house-heating boilers, and like apparatus, have interested themselves more or less in this phase of the fuel-question; but the results of their investigations are either not available, or are applicable to particular types of apparatus only.

* Prepared under the direction of L. P. Breckenridge, Director of the Engineering Experiment Station of the University of Illinois.

The first settlers in the West, coming as a rule from more eastern States, brought with them the apparatus and methods of heating and cooking with which they were already familiar. These were largely adapted to the burning of anthracite coal and wood. Throughout Illinois, until comparatively recent years, the so-called "base-burner," a stove adapted for anthracite coal only, was commonly used for heating residences of the better class. In cooking-stoves and ranges, wood or anthracite coal was, and still is, quite generally employed, especially where the expense of these fuels is not considered prohibitive. With the advent of hot-air furnaces and house-heating boilers, coming at first largely from the Eastern market, the use of anthracite was still continued to a large extent. Owing to the constantly increasing price of anthracite and to the coal-miners' strike of 1902, with the attendant scarcity of this kind of coal at that time, the use of soft coal for domestic purposes has now become much more common.

The fact that anthracite and apparatus designed for burning it were first in the field, was in itself an advantage for that fuel over Illinois or other soft coals. Anthracite possesses a comparatively high heating-value; little of it need be lost in handling; it can be burned efficiently and but a small portion of it is ash or inert matter. Consequently, a smaller weight of anthracite than of average Illinois coal will have to be handled in the generation of a given amount of heat. Moreover, anthracite is easily handled, comparatively free from dust, and has an advantage over Illinois coal in the matter of cleanliness in the boiler-room. It holds fire well. The fire is easily regulated, does not smoke or make soot to an objectionable extent, leaves little ash, and, ordinarily, the coal does not clinker badly.

The burning of Illinois coal is usually accompanied, to a greater or less extent (depending upon a number of conditions, such as furnace- and boiler-arrangements, kind, size, composition and preparation of the coal), by some or all of the following disadvantages: It is dirty, soiling clothing or other material with which it comes in contact. In handling it, more or less dust is raised. Fires are more difficult to regulate and, under many conditions, do not keep as well as an anthracite fire. Smoke, soot, and noxious gases are given off from the fire, and

these are much more apt to escape from the furnace into the boiler-room than is the case with anthracite coal. The heating-value of the coal is lower and the ash-content is higher than in anthracite, and it is difficult to burn it with the same efficiency. These conditions necessitate a larger supply for a given amount of heating to be done, more storage-space, and more handling of coal. The high ash implies a correspondingly large amount of ashes to be moved, and the tendency to clinker to a troublesome degree is more pronounced.

For the purposes of the present paper, coke can best be roughly classed with anthracite. When burned in stoves and heaters it possesses many of the properties and advantages of anthracite coal, and, to a large extent, is free from the objectionable features incident to the burning of soft coal. Like anthracite, however, it must, for use in Illinois and most other Western States, be transported long distances, or the coal from which it is prepared must be so transported. Coke as a by-product from local gas-plants is on a somewhat different footing from coke imported for fuel-purposes only, and must be considered with this fact in view.

The great advantage of Illinois coal for the Illinois user and others within a reasonable distance of the field is its low price. First-class Illinois coal for domestic purposes can be purchased for one-half or less than one-half the price of anthracite. While the heating-power of the anthracite is in general greater, the difference is not so great as to be in any sense commensurate with the difference in price. In Table I., which relates to some fuels tested under house-heating boilers at the University of Illinois, it will be noted that the B.t.u. (British thermal unit) per pound of the anthracite listed is 12,690 as compared with a value of 12,278 for a comparatively high-priced Illinois coal and a value of 10,473 for a somewhat cheaper Illinois coal. In one case the Illinois coal costs 46 per cent. of the price of the anthracite coal and contains 96.7 per cent. of its calorific capacity.

In the other case the Illinois coal costs only 34 per cent. of the price of the anthracite and contains 82.5 per cent. of its calorific capacity. This great discrepancy in price per heat-unit suggests the need of improvement in the methods of burning the cheaper fuel. Evidently, if all other conditions could be

TABLE I.—*Cost of Various Fuels.*

Fuel-tests with house-heating boilers.

Kind of Fuel.	Cost Per Ton of 2,000 Lb. at Urbana, Ill.	Cost in Per Cent. Based on Anthracite Coal as 100 Per Cent.	B.t.u. Per Lb. as Fired.	B t.u. in Per Cent. Based on Anthracite Coal as 100 Per Cent.
		Per Cent.		Per Cent.
Anthracite coal	$8.25	100	12,690	100.0
Pocahontas coal	5.50	67	14,753	116.3
Coke (gas-plant by-product)	5.00	61	12,033	94.8
Coke (Solvay process)	6.00	73	12,488	98.4
Illinois coal (Christian county), nut	2.75	34	10,473	82.5
Illinois coal (Williamson county), washed nut	3.75	46	12,278	96.7

equalized or eliminated, the B.t.u. delivered by the fuel would be the direct measure of its value.

During the past two years the Engineering Experiment Station of the University of Illinois has made many tests of different fuels, chiefly Illinois coals. Those made upon fuels burned in the furnaces of house-heating boilers of standard types and of sizes suitable for average residences have embraced anthracite and Pocahontas coal, coke, a number of Illinois coals, and briquetted coal. These tests are still going on, and will be reported in the regular bulletins of the Station, hence they will not be discussed in detail here. The relation of price to heating-value, however, will be illustrated by a few figures taken from the data at hand.

Table III. presents this relation, as based upon evaporative performance, for several of the best-known kinds of fuel. The tests were made upon two house-heating boilers, here designated as D_1 and D_2; the former, made of four horizontal cast-iron sections (the base and grate section, the fire-pot, the intermediate section, and the dome), and the latter of vertical sections, connected by means of external drums or headers. Table II. shows the dimensions.

TABLE II.—*Dimensions of Test-Boilers D_1 and D_2.*

	D_1	D_2
	Sq. Ft.	Sq. Ft.
Rated capacity, radiating-surface,	800	1,075
Area of grate-surface,	4.28	6
Sectional area of chimney,	1.07	1 07
Total heating-surface,	43.7	75.87
	Cu. Ft.	Cu. Ft.
Total water- and steam-space,	7.38	11.16

The height of the chimney above the grate in both cases is 39 feet.

Each boiler is supplied with special feed-water-supply apparatus and a load-regulator, as well as with the gauges, thermometers and other auxiliary apparatus necessary for test-purposes.

For the tests here considered, the evaporative performance of the boiler and fuel was deemed the best basis of comparison, and the tests were conducted with that as the main item sought. The problems of regulation, length of time of holding fire, smokelessness, ash, clinkers, fire-conditions, etc., were not overlooked, but necessarily became secondary in importance; and observations and results relating to these questions are not reported here.

Fires were started according to the standard method of the A. S. M. E. code. The tests varied in duration, but were approximately either 8, 16, or 24 hr. long. The fire was drawn when the boiler-pressure dropped below 4 or 5 lb. on the last firing, and did not again rise upon the opening of the damper and the closing of the check. The material drawn out at the close of the test was immediately put into a galvanized can with a close-fitting cover to prevent further combustion. Analysis of this partly-consumed or "residual" fuel furnished suitable corrections for the determination of fuel actually burned. The ash was kept separate from the residual fuel, being taken from the furnace and ash-pit before the fire was drawn. The fuel was sampled in the usual manner by taking a small portion from each firing. Analyses of the fuel, ash, and residual fuel were made at the chemical laboratories of the University of Illinois.

The feed-water, delivered to each boiler through measuring-tanks, was the condensation from heating-coils, and had a temperature near 180° F. Steam was exhausted to the atmosphere, after passing through a load-regulating device, arranged to give a load equivalent to about 65 per cent. of the boiler-rating. A separator with suitable connections was used to determine the moisture in the steam; and the usual observations concerning temperatures, pressures, drafts, etc., were made.

Table III. shows results, which are, for the most part,

TABLE III.—*Comparison of Fuel-Costs—Data and Results—Fuel-Tests with House-Heating Boilers.*

1.	2.	3.	4.	5.		6.		7.		8.		9.		10.	
Kind of Fuel.	Cost of Fuel Per Ton of 2,000 Lb. Dollars.	B.t.u. Per Lb. of Fuel as Fired.	Cost of 14,600 B.t.u. Cents.	Per Cent. of Builders' Rating Developed. (Based on 0.3 lb. water from and at 212° F., equivalent to 1 sq. ft. of radiation.)		Fuel as Fired Per Sq. Ft. of Grate-Surface Per Hour. Pounds.		Equivalent Evaporation from and at 212° F. Per Hour Per Sq. Ft. of Heating-Surface. Pounds.		Fuel-Cost of Evaporating 1,000 Lb. of water from and at 212° F. Cents.		Cost of Fuel Per 100 Sq. Ft. of Radiating-Surface served per Hour. Cents.		Efficiency of Plant (Boiler, Furnace, and Grate). Per Ct.	
				Boiler.		Boiler.		Boiler.		Boiler.		Boiler.		Boiler.	
				D_1.	D_2.	D_1.	D_2.	D_1.	D_2.	D_1.	D_2.	D_1.	D_2.	D_1.	D_2.
Anthracite coal	8.25	12,690	0.47	65.9	62.3	5.6	4.4	3.6	2.7	62.5	53.7	1.88	1.62	50.3	58.6
Pocahontas coal	5.50	14,753	0.27	63.6	64.0	5.2	4.1	3.5	2.7	40.2	32.6	1.20	0.98	44.9	55.4
Coke (gas-plant by-product)	5.00	12,033	0.30	65.4	62.5	5.3	4.2	3.6	2.7	36.3	31.5	1.09	0.95	55.6	63.6
Coke (Solvay process)	6.00	12,488	0.35	64.4	60.8	4.6	4.0	3.5	2.6	38.1	37.1	1.15	1.11	61.1	62.9
Illinois coal (Christian county) nut	2.75	10,473	0.19	63.5	62.3	7.8	7.0	3.5	2.7	30.1	28.6	0.91	0.86	42.0	44.4
Illinois coal (Williamson county) washed nut	3.75	12,278	0.22	63.9	64.8	6.0	5.5	3.5	2.8	31.2	28.7	0.93	0.86	47.4	51.5

averages of from three to six tests with each kind of fuel. Columns 1 and 2 give the kind and cost of each fuel. The samples were purchased mostly from local dealers, and the prices are given for quantities of from 1 to 5 tons only. Almost all these fuels can be purchased somewhat more cheaply in larger quantities; but domestic consumers are very likely to be retail purchasers. Column 3 gives the heating-capacity per pound for each of the fuels listed, and column 4 the cost of 14,600 B. t. u. as purchased in each. (The number of 14,600 B. t. u., as the calorific capacity of a pound of pure carbon, is taken as a convenient unit for comparison.) Column 5 shows that the boilers were operated at practically the same average capacity. The evaporation of 0.3 lb. of water per hour from and at 212° F. is taken as the equivalent of 1 sq. ft. of radiation. Columns 6 and 7 give two of the principal operating-conditions. Columns 8 and 9 give the cost of evaporating 1,000 lb. of water from and at 212° F., and the fuel-cost per hour of serving 100 sq. ft. of radiating-surface. Column 10

gives the calculated efficiency of the boiler, furnace, and grate, operated under the conditions of the tests.

It will be noted that the cost of evaporating 1,000 lb. of water from and at 212°F. varies from 62.5 cents for anthracite to 28.6 cents for the cheaper of the Illinois coals tested. If evaporative performance alone be considered, this shows a saving of 54.2 per cent. of the cost of the anthracite in producing the same effect. The similar differences between the same Illinois coal and the Pocahontas coal and the cokes, while not as great, are sufficient to warrant, other conditions being equal, the choice of Illinois coal on the ground of economy. A possible saving of from 10 to 50 per cent. in the cost of a material which enters into practically every home, and the consumption of which in Illinois alone is annually millions of tons, at several dollars per ton, is an object worthy of every possible effort.

The endeavor of manufacturers to furnish stoves and furnaces suitable for Illinois coal, and that of coal-dealers and others interested to disseminate information concerning this question, are evidence of a demand for such apparatus and information, indicating that the general public appreciates the advantage to be gained by using this coal, as soon as some of its more pronounced disadvantages have been eliminated. Indeed, by the extent to which it is already using soft coal for domestic purposes, in spite of the attendant disadvantages, the public shows that it is determined to have the cheaper fuel. In this matter, the consumer is ahead of the manufacturer, the coal-dealer, and the investigator.

VII. The Smokeless Combustion of Bituminous Coal.

By A. Bement.*

The present paper deals specially with Illinois coal; but the problem of smokeless combustion is the same for all bituminous coal, its difficulty being proportional to the amount of volatile matter in the fuel. It is to be assumed for the present purpose that coal from the Eastern Interior field, of which Illinois is a part, will make practically as much smoke as any other bituminous fuel, and, therefore, that a method or ap-

* Consulting Engineer, Chicago.

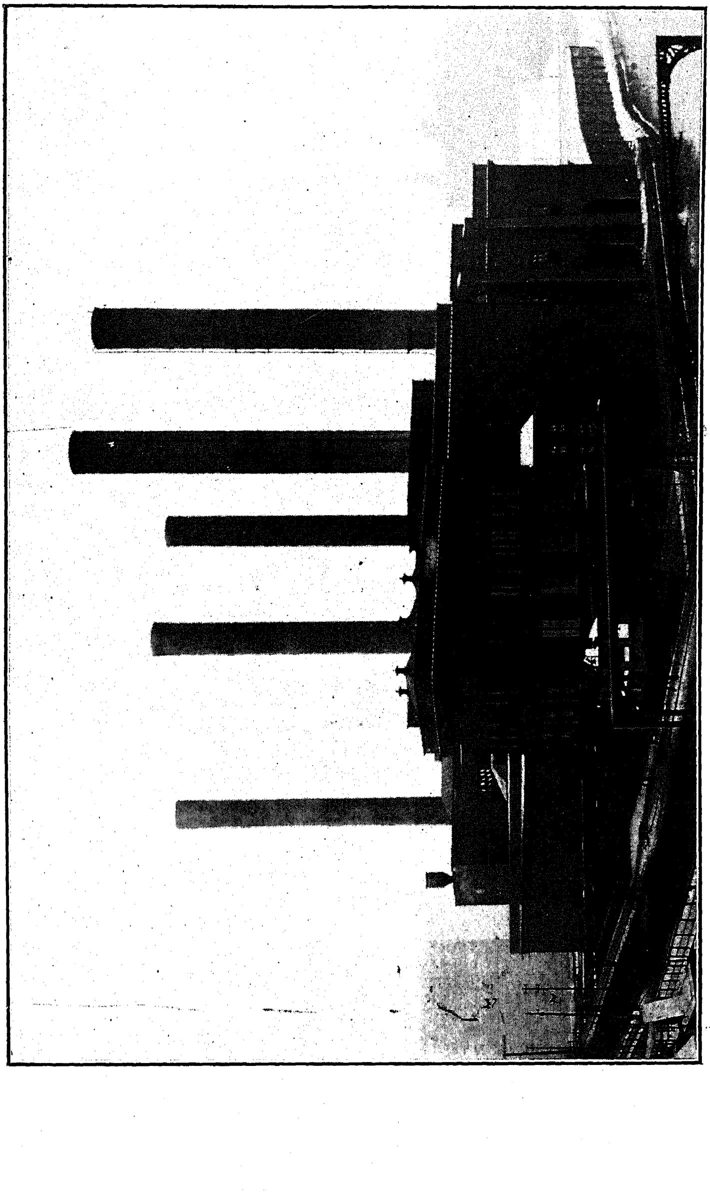
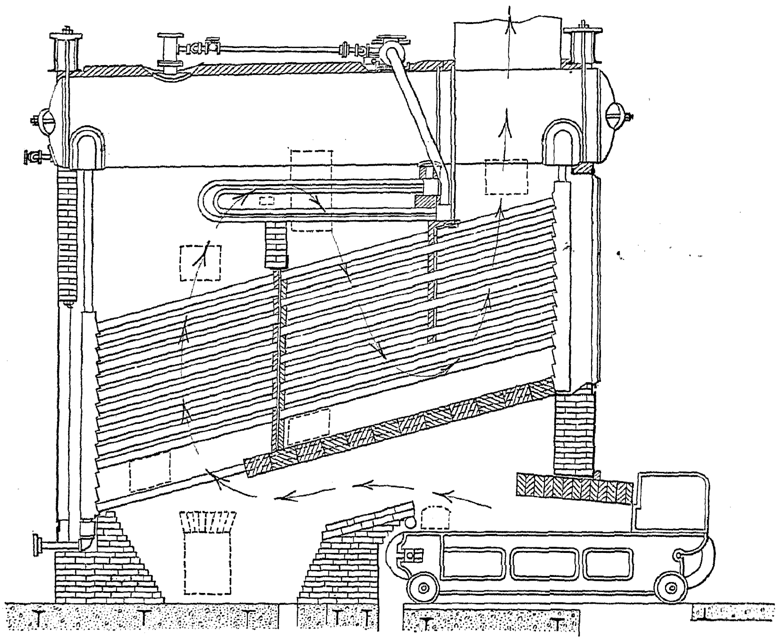


Fig. 1.—Harrison Street Electric Station, Commonwealth Edison Co. Smoke-Proof Furnace in Full Operation.

paratus adequate for the smokeless combustion of this coal will be useful for any other.

Many persons still honestly doubt the possibility of burning bituminous coal without smoke. But experience has proved that it is entirely feasible in steam-making to employ apparatus which will be, except when fires are first lighted, entirely smokeless, so that a photograph taken of the chimney would show no smoke whatever—in fact, would give no indication



LONGITUDINAL SECTION.

FIG. 2.—IMPROVED FORM OF BOILER, SERVED BY SMOKE-PROOF FURNACE OF THE KIND USED IN ELECTRIC STATION SHOWN IN FIG. 1.

whether the chimney was in service or not. Since, in many plants, fires are lighted only about once a month, this is practically a continuous smokeless operation. The achievement is due entirely to the inherent characteristics of the apparatus itself, and is not dependent in any sense upon the care or skill of the attendant. Chimneys have remained smokeless even when coal was being burned at the rate of one ton per minute.

Fig. 1 is from a photograph of the Harrison Street electric generating-station of the Commonwealth Edison Co., Chicago, at which plant originated the smoke-preventing scheme here described. The boilers are of the Heine type, served by chain-

grate stokers, and the application of the tile roof originated with W. L. Abbott, chief operating engineer of that company. The principle of the furnaces is illustrated by Fig. 2, which shows an improved form of water-tube boiler, devised by me, employing this tile furnace-roof, which is carried by the lower row of tubes in the boiler, thus forming an adequate combustion-chamber.

The requirements for smokeless combustion, simply stated, are: (1) that the evolution of gas from the coal shall proceed uniformly; (2) that the gases distilled uniformly from the coal shall enter a fire-brick chamber, either (*a*) of sufficient length to allow their complete natural combustion, or (*b*) provided with such auxiliary mixing- and baffling-devices as will effect the artificial mixture and complete combustion of the gases before their exit from the chamber.

Uniform evolution of the volatile gases of the coal is the essential feature of the process, and it is for this reason that mechanical stokers, as a class, are more effective in preventing smoke than any apparatus accompanied with intermittent firing. A stoker, however, which, through abnormal working or incorrect manipulation, feeds irregularly, has the effect of a hand-fired furnace. Hence, forms of stokers depending upon gravity-feed or having an inclined grate are objectionable, because sliding of the coal, or disturbance of the fuel-bed by the attendant, may cause fresh coal to roll down in a large mass. Again, stokers which require that the fuel-bed be disturbed in the removal of ash and clinkers cannot be depended upon for uniform feed of the coal, except under conditions of most favorable manipulation and suitable size and character of fuel. The chain-grate stoker, which operates with a horizontal fuel-bed, receiving the fresh coal at one end and automatically and continuously discharging ashes at the other, insures a uniformity in feed of coal and condition of fuel-bed not attained hitherto with any other machine of the kind. This form of stoker is shown in Fig. 2. In combination with a tile furnace-roof, it satisfies requirements (1) and (2 *a*) above stated. The adoption of this form of apparatus is extending rapidly. The University of Illinois has recently employed it in connection with an experimental boiler in its engineering laboratory.

Present practice largely tends, however, to some intermittent form of fuel-supply, such as an irregularly-working stoker, or hand-firing, and attempts to secure a smokeless combustion are generally hampered by such conditions of firing. In such cases, requirement (2 *b*), above stated, becomes imperative; and, for this purpose, resort is often had to various fire-brick walls, arches, etc., and other auxiliary mixing-devices, such as steam-jets, with or without supplementary air-supply. These schemes are never entirely successful unless there is a large and well-distributed auxiliary air-supply available in the furnace-chamber immediately after firing and while the volatilization of the coal is going on, because, after a fresh charge of coal is added, there is, for the first few moments, an evolution of volatile matter at a rate enormously larger than that of the whole remaining period between firings. Now, complete combustion requires not only a proper mixing, but a proportionately adequate supply of air. Consequently, with apparatus of any intermittent type, unless the rate of fuel-supply approximates in uniformity that of a good stoker (which means, by hand, almost continuous firing), it is necessary not only to employ some powerful auxiliary mixing-device, but also to furnish at times an extra air-supply. The latter may be done by means of a steam-jet, automatically put in service as soon as the fresh coal is added, and discontinued after the expiration of a sufficient interval.

It is thus evident, that the stoker which produces results equal to that of the chain-grate is the only one which can be depended upon, under adverse conditions, to insure a positively smokeless result, independent of the skill, favorable disposition, or fidelity of the operator. Recently, a new form of underfeed stoker had been employed in the Eastern States, which in considerable measure conforms to the chain-grate in its method of disposal of the ash and in the manner of feeding the fuel. It has met with considerable favor where semi-bituminous coal, low in ash, is used. In feed of fuel and ash-removal it resembles the well known "underfeed" and Roney types of stokers, having a fuel-bed sloping to the rear, at which point are located dumping-grates for the removal of the ash—the fuel being introduced below the fire-bed at the front. Air-supply by a forced draft entering the bed at the bottom insures

that the volatile gases will become mixed with the air to a considerable extent before they leave the surface of the fire. With favorable fuel this form of apparatus has given satisfactory results under the Babcock & Wilcox type of boiler, without a tile roof, in those cases where the boiler was set high above the fire. But thus far there has been no reason to expect that with coal high in volatile matter and containing much ash it would be possible to secure favorable results without the aid of a tile-roof furnace or its equivalent. The ash would necessarily have an important effect, because greater in quantity and sometimes readily fusible. Such large clinkers might be formed that their removal would be difficult while the fire was in action. Conditions in Illinois, so far as ash-content of the fuel is concerned, are quite serious, since the usual stoker-fuel, under present methods of preparation, contains approximately 16 per cent. of ash. It is with such fuel that the result shown in Fig. 1 has been secured. The general tendency at present is towards the abandonment of hand-fired apparatus and the correction of stoker-operation so as to insure a uniformity in fuel-feed.

One of the things which has operated seriously against the installation of many stoker-applications is the general prevalence of the fallacy that it does not pay to employ stokers in small plants. It is, however, coming to be realized that only through stokers is it feasible to obtain the uniformity of feed required, not only for smokeless burning, but for good economy.

VIII. The Weathering of Coal.

By W. F. Wheeler.*

For the past two years the Engineering Experiment Station of the University of Illinois has carried on experiments to determine the nature and extent of the chemical changes taking place in stored coal.[1] Storage, under varying conditions, has been tried in order to learn how coal may be stored

* Chemist, State Geological Survey.

[1] *Bulletin No.* 17 *of the Engineering Experiment Station of the University of Illinois*, by Prof. S. W. Parr and N. D. Hamilton, presents the results of a preliminary series of tests on small samples of Illinois coal. A historical review of the literature on weathering and spontaneous combustion, and a summary of the opinions of various authorities, are also given.

with the minimum loss by weathering. In these experiments different portions of the same sample of coal were exposed to: (1) regular weather-conditions out-of-doors; (2) dry indoor-storage at about 100° F. in boiler-room; (3) the same, except that the coal was wet thoroughly every two or three days; (4) entire and continued submersion at about 70° F. Only the calorific value of the ash- and water-free coal was made use of in determining the extent of the weathering. With this factor determined for the fresh coal as a basis of comparison, the submerged coal was found to remain practically unchanged for a period of nine months, while the other three portions of the same samples showed losses varying from 2 to 10 per cent., with no marked advantage in favor of either the outdoor or indoor storage, except that the coal with a large amount of pyrite was not broken up so much when kept dry as when wet often. The loss in calorific value practically ceased by the end of five months, although a slight loss occurred during the next four months.

A new series of experiments is now going on under more nearly normal storage-conditions. Car-lot samples were obtained from three Illinois mines working different seams of coal. A car of 1.25-in. screenings and a car of 1.25-in. to 3-in. nut was shipped from each mine. One-half of each car was piled out-of-doors in an uncovered bin about 3.5 ft. deep; the other half was piled about 5 ft. deep in a covered bin, and a representative sample of each was submerged under water. Each of these cars of coal was sampled at the mine as the car was loaded, and again, about a week later, when it was unloaded. The purpose of sampling and analyzing the coal immediately after mining was to find out the composition of the coal before it had any chance to oxidize or lose its occluded gases. The second analysis, at the end of a week, was to serve as an indication of the rate of loss for that period. In Table I. the analyses of the coal up to the end of the sixth month of storage are presented.

The losses represented in Table I. range from 0.4 to 1.3 per cent. at the end of one week after mining, from 0.2 to 2.2 per cent. at the end of two months, and from 0.7 to 3.0 per cent. at the end of six months. The average loss at the end of one week was 0.8 per cent.; at the end of two months, 1.3 per cent.; and at the end of six months, 2.0 per cent.

TABLE I.—*Loss in Calorific Value During Transit and Six Months' Storage.*

Screenings.

Coal from	Sampled.	Dry Coal.			B.t.u. of Ash-, Water- and Sulphur-Free Coal.	Decrease.	
		Ash.	Sulphur	B.t.u.		B.t.u.	Per Ct.
		Per Ct.	Per Ct.				
West-ville, Illinois.	Same day as mined........	17.88	2.35	11,937	14,684	....	....
	7 days after mining........	13.84	2.58	12,462	14,627	57	0.39
	2 months after mining [a] ..	15.21	2.72	12,068	14,392	292	1.99
	2 months after mining [b] ..	15.26	2.51	12,124	14,453	231	1.57
	6 months after mining [a] ..	15.63	2.44	11,969	14,328	356	2.43
	6 months after mining [b] ..	14.51	2.25	12,081	14,247	437	2.98
	6 months after mining [c] ..	13.87	2.32	12,270	14,379	305	2.08
Spring-field, Illinois.	Same day as mined........	17.13	4.92	11,752	14,478	....	....
	4 days after mining........	17.04	4.47	11,684	14,351	127	0.88
	2 months after mining [a] ..	17.22	5.00	11,645	14,365	113	0.78
	2 months after mining [b] ..	18.33	4.70	11,414	14,254	224	1.55
	6 months after mining [a] ..	17.02	4.54	11,526	14,154	324	2.24
	6 months after mining [b] ..	17.30	4.67	11,466	14,136	342	2.36
	6 months after mining [c] ..	19.86	5.60	11,127	14,220	258	1.77
Herrin, Illinois.	Same day as mined........	14.13	3.17	12,426	14,658	....	....
	6 days after mining........	14.37	3.34	12,287	14,553	105	0.72
	2 months after mining [a] ..	15.66	2.67	12,133	14,545	113	0.77
	2 months after mining [b] ..	12.62	2.98	12,608	14,602	56	0.38
	6 months after mining [a] ..	13.76	2.84	12,342	14,476	182	1.24
	6 months after mining [b] ..	13.60	3.03	12,372	14,496	162	1.11
	6 months after mining [c] ..	14.38	3.54	13,262	14,528	130	0.89

[a] Outdoor storage. [b] Covered bins. [c] Stored under water.

3-Inch Nut Coal.

West-ville, Illinois.	Same day as mined........	10.55	4.25	12,991	14,768	....	....
	7 days after mining........	13.98	2.65	12,412	14,586	182	1.23
	2 months after mining [a] ..	14.21	2.47	12,265	14,439	329	2.23
	2 months after mining [b] ..	13.08	2.13	12,475	14,523	245	1.66
	6 months after mining [a] ..	13.53	2.10	12,396	14,456	312	2.11
	6 months after mining [b] ..	11.76	2.14	12,571	14,365	403	2.73
	6 months after mining [c] ..	15.37	3.34	12,013	14,391	377	2.55
Spring-field, Illinois.	Same day as mined........	17.87	5.75	11,741	14,655	....	....
	4 days after mining........	16.63	5.10	11,800	14,461	194	1.32
	2 months after mining [a] ..	17.45	4.66	11,626	14,361	294	2.01
	2 months after mining [b] ..	16.83	5.02	11,796	14,452	203	1.39
	6 months after mining [a] ..	16.03	4.91	11,798	14,338	317	2.16
	6 months after mining [b] ..	16.30	4.52	11,682	14,218	437	2.98
	6 months after mining [c] ..	15.90	4.21	11,854	14,338	317	2.16
Herrin, Illinois.	Same day as mined........	13.98	3.75	12,499	14,751	....	....
	6 days after mining........	14.90	3.02	12,341	14,682	69	0.47
	2 months after mining [a] ..	14.32	4.12	12,409	14,727	24	0.16
	2 months after mining [b] ..	14.08	3.84	12,378	14,634	117	0.79
	6 months after mining [a] ..	13.81	3.45	12,455	14,652	99	0.67
	6 months after mining [b] ..	13.06	3.60	12,469	14,551	200	1.36
	6 months after mining [c] ..	15.65	3.12	12,097	14,528	223	1.51

[a] Outdoor storage. [b] Covered bins. [c] Stored under water.

The loss taking place during the first week after mining represents two-thirds of the total for two months; and since very little coal can be used inside of a week after it is mined, not much importance attaches to the small additional losses of a two months' storage-period. It may be of interest to add that the amount of loss by each of the three coals in question is in line with their reputation in the market—the Herrin coal breaks up least, Springfield next, and Westville most. A small sample taken from each car is being used to determine any change in weight which may accompany the change in calorific value, but, as yet, no results are available from these experiments. It is thought, however, that the weight of the dry coal will increase, due to the addition of oxygen to the coal. A sizing-test of the coal is to be made in connection with this series of experiments to determine the amount of disintegration which takes place. It seems probable that this change in the size of the coal may have greater economic importance than the slight change in its composition. The loss in calorific value shown by the car-lots of coal in a period of two months is smaller than was expected, judging from the results of the preliminary series of experiments. This discrepancy is accounted for, most likely, by the greater exposure of the early samples, which consisted of but 25 lb. of coal each.

Two samples of old pillar-coal were also collected and compared by analysis with fresh coal from the same mine, to determine the extent of weathering in coal exposed for a long time underground. As the analyses in Table II. show, the loss in calorific value is not very great, being in both instances under 3 per cent.

The Edwards coal presents an extreme case of weathering. The second sample was taken from near an outcrop that had been covered with soil and forest on a gentle slope, and had not been subject to erosion in recent years. The coal in this case had become so changed as to appear nearly like lignite, and the analysis shows a corresponding resemblance. The high moisture, nearly 30 per cent., is characteristic of the lignites, as is also a high percentage of oxygen, and a low calorific value.

Another point of importance in connection with the weathering of coals of the type found in Illinois and our Western States is the occurrence of a loss in calorific value even when samples are kept sealed up away from the air. The coal loses

TABLE II.—*Analyses of Pillar-Coal and Fresh Coal.*

Coal from		Total Moisture.	Analysis of Dry Coal. Ash.	Volatile Matter.	Fixed Carbon.	Sulphur.	Heat.	Heat of Ash-, Water- and Sulphur-Free Coal.
		Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	B.t.u.	B.t.u.
Belleville, Ill.	Pillar-coal exposed 22 years.............	10.18	16.21	38.26	45.43	5.01	11,797	14,472
	Fresh face, same mine.......	9.76	15.80	41.29	42.91	4.76	12,202	14,785
Equality, Ill.	Pillar-coal exposed 27 years.............	4.76	13.84	36.56	49.60	3.84	12,514	14,754
	Fresh face, same mine.......	4.47	10.85	47.82	51.33	3.72	13,235	15,188
Edwards, Ill.	Fresh face 300 ft. from outcrop..........	13.86	16.25	40.72	43.03	3.91	12,044	14,618
	Outcrop-sample...............	29.81	16.86	39.27	43.87	0.85	9,257	11,164

methane at first and absorbs oxygen from whatever air may be in contact with it. Each of these processes accounts for a part of the loss of heat-units. If umpire-samples are to be kept in connection with coal-contracts calling for a specified calorific value, the "heat-unit basis," this fact must be kept in mind, since the loss in this way may become as much as 300 B.t.u. in a few months, as was shown in connection with my paper[1] presented at the Toronto Meeting of the Institute, July, 1907, and again, with additional data, in the *Journal of the American Chemical Society*, for June, 1908.

Summary.

The results to date on this series of tests confirm the conclusion set forth in the summary of the *Bulletin* by Prof. S. W. Parr and Mr. Hamilton,[2] except that 4 per cent. seems to be amply sufficient to cover the losses sustained by Illinois coals under regular storage-conditions, the larger losses indicated in the former series being probably due to the small size of the samples exposed as against car-load lots in the present series. In these latter tests, the losses sustained by the submerged coal, though small in amount, are only slightly less than those indicated for the exposed coal.

[1] Pure Coal as a Basis for the Comparison of Bituminous Coals, *Trans.*, xxxviii., 621 to 632 (1908).

[2] *Bulletin No.* 17, Engineering Experiment Station of University of Illinois (1907).

IX. The Modification of Coal by Low-Temperature Distillation.

By C. K. Francis.*

Since 1902 the laboratory of applied chemistry of the University of Illinois, under the direction of Prof. S. W. Parr, has been engaged in the investigation of bituminous coal, especially from Illinois, with a view to such a modification of it as will permit combustion under ordinary conditions without the production of smoke. Recent investigation, directed primarily to the development of fundamental facts and principles, has included a careful study of the chemical changes or reactions that may accompany the treatment of coal under varying temperatures and in different atmospheres.

Briefly outlined, the method is as follows: About 4 or 5 lb. of the coal was placed in a cylindrical retort, fitted with a ¾-in. pipe at each end, one pipe serving as inlet-tube for the gas used as an atmosphere, and the other as an outlet for the gases produced. These pipes also permitted the revolving of the retort during the operation. The atmospheres used in the experiments so far have been nitrogen, oxygen, and steam. After the air was washed out of the retort by the gas to be experimented with, the retort was heated. The period of heating varied from 2 to 8 hr., the temperature from 200° to 425° C.

Nitrogen.—The results obtained in an atmosphere of nitrogen may be represented by those from a test in which the coal was heated for 3 hr., at an average temperature of 402° C. The analyses of the coal and the product, calculated for the same amount of ash, dry basis, are given in Table I.

Table I.—*Analyses of Coal and Product.*

	Original Coal. Per Cent.	Product. Per Cent.
Ash,	8.30	8.30
Volatile matter,	36.23	15.26
Fixed carbon,	55.47	51.98
Sulphur,	2.24	1.32
B.t.u.,	13,244	9,819
B.t.u. (unit coal),	14,567	14,702

The expression "unit coal"[1] is used here to represent the

* Engineering Fellow in Applied Chemistry, University of Illinois.

[1] *Journal of the American Chemical Society*, vol. xxviii., No. 6, p. 632 (May, 1906); and *Trans.*, xxxviii., 621 (1908).

ash-, water- and sulphur-free material. The formula for calculating the B.t.u. to the "unit-coal" basis may be expressed as follows:

$$\frac{\text{B.t.u.} - (\text{Weight of S} \times 4{,}050)}{100 - (\text{Ash} + H_2O + \frac{5}{8}\text{ S})} \times 100 = \text{B.t.u. per lb. unit coal.}$$

The gas evolved from the coal under this treatment had the following composition:

	Per Cent.
Carbon dioxide and hydrogen sulphide,	17.33
Illuminants,	9.54
Oxygen,	0.0
Carbon monoxide,	7.66
Methane,	32.66
Hydrogen,	2.37
Nitrogen,	29.97
Volume of gas,	50 liters.
Weight of coal,	2,120 g.

Steam.—The results after treatment in an atmosphere of steam for 2 hr. at an average temperature of 381° C. were as follows:

	Original Coal. Per Cent.	Product. Per Cent.
Ash,	8.72	8.72
Volatile matter,	38.07	25.78
Fixed carbon,	53.19	55.79
Sulphur,	2.57	2.14
B.t.u.,	13,304	11,959
B.t.u. (unit coal),	14,605	14,813

The gas evolved from the coal under this treatment had the following composition:

	Per Cent.
Carbon dioxide and hydrogen sulphide,	32.40
Illuminants,	7.30
Oxygen,	0.80
Carbon monoxide,	9.60
Methane,	20.60
Hydrogen,	0.00
Nitrogen,	29.30
Volume of gas,	37 liters.
Weight of coal,	2,400 g.

The carbon dioxide present in this gas, and, probably, also that produced when nitrogen was the atmosphere, may have

been due to the residual oxygen in the retort. Indeed, the decrease in quantity of the carbon dioxide as the process was continued seems to indicate some such explanation.

Oxygen.—The results after treatment in an atmosphere of oxygen for 4.5 hr. at an average temperature of 379° C. were as follows:

	Original Coal. Per Cent.	Product. Per Cent.
Ash,	8.52	8.52
Volatile matter,	37.42	18.72
Fixed carbon,	54.05	60.20
Sulphur,	2.16	1.71
B.t.u.,	13,399	11,588
B.t.u. (unit coal),	14,768	14,793

The composition of the gas evolved from the coal under this treatment was as follows:

	Per Cent.
Carbon dioxide and hydrogen sulphide,	12.73
Illuminants,	3.53
Oxygen,	9.27
Carbon monoxide,	4.74
Methane,	13.68
Hydrogen,	0.00
Nitrogen,	56.05
Volume of gas,	50 liters.
Weight of coal,	2,000 g.

In all cases the product has a lower heat-value than the coal. This reduction is accounted for by the hydrocarbon values represented in the gaseous and oil-products of distillation. Especial attention should be given to the heat-values calculated to the unit-coal basis. These values show a consistent increase throughout. A tentative explanation is, that the oxygen and nitrogen compounds of the volatile matter have been more largely driven off than the hydrocarbon compounds. If the loss in volatile matter, as shown, had been chiefly that of the marsh-gas (CH_4) series, a reduction in heat-values for unit coal must result. If, however, the loss is made up of water of composition, there would be a relative increase in the heat-value of the residual coal. The weight of water condensing in the flasks and separated from the oil, showed in each test an excess over the possible amount which could come from the free water

present, amounting to 3 per cent. in Test No. 4, 4.5 per cent. in Test No. 6, and a little less than 3 per cent. in Test No. 7. These figures must represent the percentage of decrease in the water of composition. A loss of 2 per cent. in this constituent would raise the B.t.u. factor, referred to the unit-coal basis, from 14,567 to 14,864. This seems to warrant the conclusion that a loss of water of composition occurs, which is an important point for further confirmation, since a fundamental purpose of this investigation is to develop, as nearly as may be, the conditions which govern the various decomposition-processes.

Enough has already been developed to indicate that the product obtained by the treatment here outlined, or possibly a combination of two atmospheres, would have a special value for domestic use and for such industrial operations as require a smokeless fuel. While much of the volatile constituent remains, it has undergone a change which makes it not difficult to carry on combustion without the production of smoke. This fact is, perhaps, suggested by the rather close resemblance in composition to the so-called smokeless coals. Because of the very fragile character of this material, it would need probably to be briquetted.

The investigation of certain phenomena noticed in the preliminary experiments and in the work just described, suggested certain specific investigations. For example, carbon dioxide was present in the evolved gases when an inert gas, nitrogen or steam, was used as an atmosphere. In each case the amount was considerable, ranging from 12 to 27 per cent. In several of the tests an occasional rise of the temperature in the retort was noted, seemingly independent of the internal source of heat. The first investigation suggested was the determination of the temperature at which oxidation of coal begins, and the actual ignition-point in different atmospheres. The apparatus devised for this purpose consisted of a purifying-train, a heating-chamber, and an apparatus for detecting carbon dioxide when evolved. In the flask employed as a heating-chamber were placed two thermometers, one of which indicated the temperature of the gas, the other that of the coal under observation. Any difference in the readings of the two was due to reactions taking place within the coal. Oxidation was said to begin when carbon dioxide was detected at the outlet.

The results of these tests may be summarized as follows: Finely-pulverized coals in contact with oxygen, either pure or diluted, as in the case of air, begin to oxidize at between 120° and 135° C. In some instances, however, this temperature of oxidation is higher, but in none of the tests did it exceed 155° C. The ignition-temperature varies with the type of coal and, to a certain extent, also with the fineness of division. Powdered bituminous coals ignite in oxygen at a temperature of about 160°; buckwheat sizes ignite at from 260° to 300°; finely-divided semi-bituminous coals at about 200°; and anthracite at about 300° C. Bituminous coals ignite in air at about 330° C.

The investigations of the phenomena occurring under the same conditions in atmospheres of steam and nitrogen are not completed. It has been demonstrated that no appreciable amount of carbon dioxide is formed in an atmosphere of pure steam, but at 315° C. there is an abrupt rise of temperature in the coal of over 50°, the limit of the thermometer preventing an exact determination. Since no increased appearance of carbon dioxide accompanied this rise in temperature, it must be attributed to the exothermic character of the decompositions occurring at that stage. Similar conditions were observed in a corresponding experiment, using nitrogen; but, since a small amount of oxygen remained in the nitrogen, giving as a result a moderate test for carbon dioxide at the exit-tube, this matter of temperature-differences in nitrogen must await further and more careful examination. Indeed, the general proposition here indicated, of a probable exothermic behavior, is of considerable importance, and calls for a carefully-devised series of experiments, which are now in progress.

X. Summary and Conclusions.

By H. Foster Bain.

Coal-Reserves.—In the section of this paper written by Mr. DeWolf, the amount of available coal in Illinois is shown to be so large as to warrant a feeling of security for the future. It is not likely that any scarcity will be felt in this field for several generations to come. While, as indicated by Mr.

Lindgren in the analyses submitted, the grade of the coal is not the highest, this is more than offset by the abundance.

Mining-Costs and Conditions.—It has been shown by Mr. Rice that there are large losses in the present methods of mining employed in this field, but that the remedy for these may be easily found so far as technical performance is concerned. The difficulties lie in the financial and industrial situation, and until the average price of coal increases or the rate of interest falls, only minor improvements are to be expected. Certain changes which are possible even under present conditions are pointed out.

Present Methods of Utilization.—Illinois coal is now largely used for domestic heating, power-generation and locomotive consumption. For the first purpose the most important limiting-factor is its supposed lower heating-value as compared with competing Eastern coals. This, Mr. Snodgrass shows, is, price considered, fictitious; and it is suggested that with the development of proper apparatus such actual difference as does occur may be decreased, if not entirely wiped out. It is therefore to be expected that the domestic market for Illinois coal will in the future not only expand with the population but at an increasing rate, both by actually displacing competing coals and by the capture of a larger share of the new markets. While this phase of the subject has not been discussed here, it may be pointed out that the increased use of washeries in preparing coal for the market will aid this movement by furnishing to domestic users a cleaner and lower-ash coal.

For power-generation in stationary plants a smokeless coal is becoming increasingly important, and Mr. Bement shows that it is entirely possible to burn the worst of the Illinois coals with extremely satisfactory results as regards smoke. This removes one of the large handicaps under which Middle Western coal has heretofore labored. One of the developments of the future will undoubtedly be towards larger central power-plants and the distribution of energy, probably as electricity.

One other development which is likely to influence the future market of Illinois coal is the larger use of the gas-engine. It is to be regretted that specific data on this point could not be included in this symposium. The reports of the

Fuel-Testing Plant of the U. S. Geological Survey[1] show that Illinois coals are excellently adapted to such use; and as a matter of fact they are now being so used at one or two points. The large installation of gas-engines in the steel-plant at Gary is one of the significant signs of the times. It cannot be doubted that there will be an increasing use of gas-engines; and since the gas-producer tends to some extent to wipe out the margin between high-grade and low-grade coals, in the long run this change will be to the advantage of the coal-fields of the interior.

It is somewhat difficult to get at the amount of coal used for locomotive purposes. In the fiscal year ending June 30, 1907, Illinois produced 46,700,608 tons of coal. For the corresponding term the locomotive consumption for the State of Illinois, as given by the State Railway and Warehouse Commission, was as follows:

Locomotive Consumption of Coal in Illinois.

	Passenger Service.	Freight Service.	All Service, Including Switching and Construction.
Miles run,	39,183,431	50,167,137	113,584,275
Tons used,	2,140,199	4,590,916	9,220,119
Pounds per mile,	109.34	183	138.01

These figures are representative only. While most of the coal came from the Illinois mines, a minor portion was from Indiana and Eastern States. On the other hand, a very large tonnage of Illinois coal goes to supply locomotives running in other States. At present it is impossible to give figures covering this tonnage, though they are being collected.

The performance-figures given, representing as they do a very large engine-mileage, may safely be assumed as averages in computing future consumption with increased railway-activity. The average number of engine-miles per ton, including switching and all kinds of service, amounted to 12.85. Corresponding figures for September, 1907, for various roads in the Middle West using Illinois coal were, 13.38, 12.2, 14.7, 17.18, 13.69. The figure is probably a little low rather than the reverse. It will be noticed that approximately 20 per cent.

[1] *Bulletins* Nos. 261 (1905), 290 (1906), 332 (1908), and *Professional Paper* No. 48, *U. S. Geological Survey*, pp. 981 to 1325 (1906).

of the coal-output was used for locomotive purposes in the State. In 1905 the corresponding figure was 25 per cent. About half of this is used in the freight service, and it may be roughly computed that 1,000,000 tons are burned in hauling to place of consumption the remaining 50,000,000 tons of output.

Possible Future Improvements.—One great bar to a wide use of Illinois coals is the poor shipping-quality. They do not stand rehandling, and in storage they are subject to deterioration and spontaneous combustion. For this reason they do not enter distant markets, except by all-rail routes, and at the season of maximum demand. This not only limits the total output of the mines, but adds to the normal cost per ton the expense of idle plants and men for about 30 or 40 per cent. of the time, it being necessary in order to meet maximum demands to have a capacity in large excess of the average demand. For these reasons studies of the weathering and deterioration of our coals are of peculiar interest. Different phases of this subject are brought out by Messrs. Barker and Wheeler, while Mr. Francis gives some of the fundamental facts regarding the decomposition of coal and its oxidation at low temperature. The weathering studies now going on are too incomplete to permit the drawing of general conclusions: It is shown that submerged coal does not deteriorate, except by loss of the occluded gas; and, since it is also protected from spontaneous combustion, such a storage-plant, where commercially feasible, may be best. It is hoped, however, that a more satisfactory solution of the difficulty may be found. In this connection Mr. Francis's work on what might be called "anthracitizing coal" offers some suggestions, though not as yet any conclusions commercially available.

Markets for Illinois Coals.—The coal-production and consumption for 1906, the last year for which figures are available, may be estimated as shown in the tabulated statement on the following page.

It will be noted that the railways are the largest users of Illinois coals. Next to them stand the cities of St. Louis and Chicago. On the face of the figures St. Louis is much the larger user. These figures are slightly deceptive, since it is impossible to separate the Eastern coal, aside from anthracite, from the Illinois coal handled by the roads running into St. Louis.

Production and Consumption of Illinois Coal for 1906.

	Tons.	Tons.
Total production.	41,480,104	
Consumption :		
Used by railways within the State (estimated by percentage),		9,333,000
Shipped to Chicago,		4,968,102[a]
Shipped to St. Louis,		6,600,216[b]
Used at mines,		1,374,308[c]
Sold to local trade and to employees,		2,891,220[c]
Shipped to railways outside, and to local consumers within and outside the State,		16,313,158[d]
Total,		41,480,104

[a] Chicago Bureau of Coal Statistics.

[b] St. Louis Coal Traffic Bureau. These figures include Eastern coal received by railway.

[c] According to U. S. Geological Survey.

[d] By difference, assuming no stock carried over.

In 1906, in Chicago, 2,961,926 tons of Indiana coal were also used. In the year studied 4,265,528 tons were used at or near the mines and did not enter the general competitive market. It is, apparently, safe to conclude that at least half the coal mined is used within the State. If the quantity of coal shipped from the mines to local dealers were known it is probable that this portion would be found to be larger. In other words, Illinois derives not only the benefit of mining but the profit from burning half its output.

Of the local markets, that of Chicago is the most important and can be studied in most detail. The receipts and shipments at this point in 1906 are given below.

Coal-Receipts and Shipments at Chicago in 1906.

(Chicago Bureau of Coal Statistics.)

Receipts.	Tons.	Tons.
Anthracite :		
By lake,	781,751	
By rail,	744,531	
Stock, Jan. 1st,	403,976	
Total,		1,930,258
Pennsylvania bituminous,	925,237	
Ohio coal,	856,833	
West Virginia,	914,420	
Coke,	342,919	
Total Eastern bituminous and coke,		3,039,409

Receipts.	Tons.		Tons.
Illinois coal:			
Northern field,	1,100,915		
Southern field,	3,153,956		
Central field,	270,456		
Eastern field,	442,775		
Total Illinois,		4,968,102	
Indiana coal:			
Brazil block,	165,075		
Miscellaneous bituminous,	2,796,851		
Total Indiana,		2,961,926	
Total Western coal,			7,930,028
Total coal and coke,			12,495,719
Shipments:			
Anthracite,			542,554
Bituminous,			2,772,204
Coke,			258,316
Total shipments,			3,573,164

Corresponding figures for a series of years show that in this important market the use of Illinois and Indiana coal has been increasing at the expense of Eastern coals. That at the present ratio of prices they may well be expected to continue to do so, is shown by the following tabulation, in which the Chicago price is an average of the *Black Diamond* weekly quotations for January and February of 1907 (a period of normal demand), and the fuel-value is based on large commercial deliveries for one year with semi-monthly sampling and analysis by the Fuel Engineering Co. For convenience in further discussion the freight-rate to Chicago is added. While it is probable that only a portion of the coal marketed in Chicago actually sold at the prices indicated, much being delivered under long-time contracts, the prices none the less fix the ratio of competition, since they are the quoted prices for the excess coal reaching out for a market.

Competing Coals in Chicago Markets.

Coal.	Chicago Price.	Freight-Rate.	Thousand B.t.u. for a Cent.
Franklin county (Ill.) screenings	$1.59	$1.00	140
Clinton county (Ind.) mine-run	1.77	0.70	124
Springfield (Ill.) mine-run	1.75	0.75	122
Pittsburg (No. 8 Ohio)	2.91	1.60	90
Carterville, Ill., washed No. 2	2.90	1.00	80
Pocahontas	3.46	2.05	80
Hocking (Ohio)	3.54	1.65	70
Indiana block	3.20	0.80	68

The average for the five Western coals gives 107,000 B.t.u. for a cent, and for three Eastern coals, 80,000 B.t.u. It will be noted that consumers using Eastern coals and Indiana block pay on an average 64 per cent. greater coal-bills as a penalty for not adapting their furnaces to the burning of the local coals. At these prices the producer of Carterville washed coal is able to spend 49 cents a ton more on his product than the miner of Pocahontas coal, and yet deliver the same number of heat-units for a dollar to his Chicago customers. It is evident that Pocahontas, the best of the Eastern coals, can never be delivered in Chicago at existing freight-rates in competition with Illinois coals when consumers adapt their furnaces to economical and smokeless burning of the latter.

This leads to an inquiry regarding existing and future freight-rates. As is well known, coal freight-rates generally are very low and the rate per mile decreases rapidly with distance. The following are a few coal freight-rates per ton per mile.

Illinois Coal Freight-Rates.

	Rate. Cents.	Approximate Rate Per Ton-Mile. Cents.
Northern Illinois to Chicago,	50	0.50
Central Illinois to Chicago,	75	0.37
Southern Illinois to Chicago,	100	0.33
Chicago and Northern Illinois to St. Paul and Minneapolis,	140	0.28
Central Illinois to St. Paul and Minneapolis,	180	0.28
Southern Illinois to St. Paul and Minneapolis,	210	0.29
Peoria to Omaha,	226	0.54
Central Illinois to Omaha,	202	0.28
Southern Illinois to Omaha,	237	0.25
Pittsburg district to Chicago,	165	0.33
Pocahontas to Chicago,	205	0.30

While these figures show some difference between local and long-distance shipments it must be remembered that a minimum initial-charge is to be taken into account, and if they be plotted it will be seen that there is small likelihood of the local rates decreasing much relative to the distant rates unless the whole system be changed. It is to be expected, therefore, that, so far as freight-rates are concerned, competition will remain substantially as it is at present.

The decrease in the rate per ton per mile with distance gives

a marked advantage to higher-grade coals in the distant market. If, for example, two coals differ in original price 25 cents, selling for $1 and $1.25 a ton respectively, the difference is equivalent to 20 per cent. of the price of the better coal. If now an initial freight-rate of 50 cents be paid, the difference amounts to only 14 per cent., and with each increase in freight the original difference in cost decreases until when a $2 rate is paid it amounts to only 7.6 per cent. This is the explanation of the fact that screenings are used near the mine, and only washed coal and lump are exported to a distance. Indeed, lump-coal has a fictitious value within the mining-regions and finds a local market for special purposes only.

Practically no Illinois coal moves eastward. This is due not only to competition based on the quality of the Eastern coals, but to the present organization of freight traffic, which makes it difficult to get cars. Such coal as goes east from this coal-field is supplied by Indiana. To the west, Illinois coal dominates the markets of Missouri and Iowa almost to the eastern margin of their own coal-fields, and has a scattering trade beyond. To the southwest, coal is furnished to the railways to a point about half way between St. Louis and Kansas City, and to a few supply-stations beyond. Directly south, there is very little coal-movement except to supply certain connecting railways. The larger markets are dominated by Eastern coal shipped by river, a traffic practically closed to Illinois operators for the present, owing to lack of terminals within the State, and the poor stocking-qualities of the coal. To all intents and purposes the only Illinois coal delivered to the rivers is that used by the local steamboats.

To the north and west, the coal goes in large quantities into southwestern Wisconsin, northern Iowa, southern Minnesota, and eastern South Dakota. On the one hand it must meet the competition of the nearer Iowa fields, and on the other, of the lake-shipped Eastern coal. The size of this lake trade may be illustrated by the figures for 1907.

Lake-Shipments of Coal in 1907.

	Tons.
Western Pennsylvania coal,	8,306,143
Ohio coal,	3,703,322
West Virginia coal,	3,343,752
Total,	15,353,217

The lake coal dominates the market as far south as Milwaukee, and it is only of recent years that Illinois coal has begun to go in any quantity as far northwest as St. Paul and Minneapolis. In the territory between these points there is much debatable ground, and if methods of storage can be devised so that the coal may be shipped in the summer, large increases in trade may be expected. The same is true of western Iowa and eastern Nebraska, where at present there is only a moderate trade. If, in addition to finding a solution of the storage-problem, water-transportation be made available, Illinois coal may become a dominant factor in the Northwest. It must be admitted, however, that this is far from being accomplished, and for the present, in extending the markets, reliance must be placed mainly on a campaign of education in the proper burning of high-volatile coals.

The purchase of coal on specifications is also to be commended. This not only leads to closer studies of coal-bills and conditions of burning, but, by means of the inspection-system, improves the mining and cleaning of the coal. While doubtless criticism can be fairly made of particular specifications, it is believed that the system itself will, in the long run, commend itself both to buyers and sellers.

The Clinton Iron-Ore Deposits in Alabama.*

BY ERNEST F. BURCHARD, WASHINGTON, D. C.

(Chattanooga Meeting, October, 1908.)

CONTENTS.

I. INTRODUCTION.

BRIEF and interrupted studies of the Alabama Clinton iron-ores have been made within the last four years by members of the U. S. Geological Survey. Reports on the progress of this

* Published by permission of the Director of the U. S. Geological Survey.

work have been published from time to time by the Survey.[1] A detailed report on the Birmingham district, with maps, has been completed, and will be published within the next year.[2]

In the following paper it is aimed to present only an outline of the geologic relations of the ores, since the Appalachian geology of Alabama has been discussed by many previous writers, but to describe rather fully the ore of the Birmingham district, and to discuss its relations and probable extent.

II. Outline of the Geology.

1. *Stratigraphy.*

The pre-Pennsylvanian rocks exposed in and along the borders of the valley-regions of Alabama, range from middle Cambrian to Mississippian (Lower Carboniferous), and at the SW. extremities of the valleys these rocks are overlain by coastal-plain deposits of Cretaceous and Tertiary age. A generalized section of the pre-Pennsylvanian Palæozoic rocks for the valley-regions is given in Table I.

Table I.—*Section of Pre-Pennsylvanian Palæozoic Rocks in Valley-Regions of Alabama.*

System.	Formation and Character.	Thickness.
		Ft.
Carboniferous (Mississippian)	Shale and sandstone	300 to 2,200
	Limestone and shale	300 to 800
	Sandstone and shale	50 to 200
	Fort Payne chert and limestone	100 to 300
Devonian	Chattanooga shale	0 to 30
Silurian	Clinton ("Rockwood") shale, sandstone, and iron-ore	200 to 700
Ordovician	Chickamauga ("Trenton") limestone	200 to 800
Cambro-Ordovician	Knox: dolomite and chert	2,200 to 2,700
	Knox: basal dolomite	500 to 600
Cambrian	Conasauga shale and limestone	1,000[a] to 1,500

[a] Base not exposed.

[1] Burchard, E. F. Iron Ores in the Brookwood Quadrangle, Ala., *Bulletin of the U. S. Geological Survey* No. 260, pp. 321 to 334 (1905).

Eckel, E. C. The Clinton or Red Ores of Northern Alabama, *Bulletin of the U. S. Geological Survey* No. 285, pp. 172 to 179 (1906).

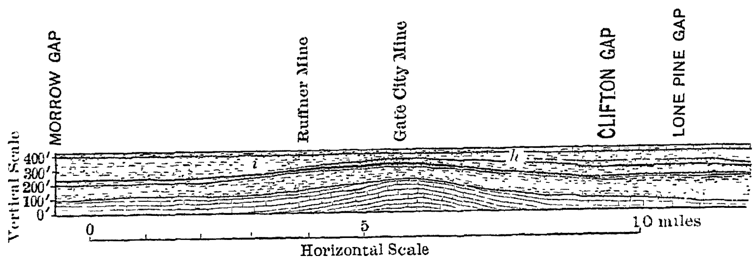
Burchard, E. F. The Clinton or Red Ores of the Birmingham District, Ala., *Bulletin of the U. S. Geological Survey* No. 315, pp. 130 to 151 (1907).

Burchard, E. F. An Estimate of the Tonnage of Available Clinton Iron Ore in the Birmingham District, Ala., *Bulletin of the U. S. Geological Survey* No. 340, pp. 308 to 317 (1908).

[2] Eckel, Edwin C., Burchard, E. F., and Butts, Charles. The Iron-Ores, Fuels, and Fluxes of the Birmingham District, Ala., *Bulletin of the U. S. Geological Survey* No. — (in preparation).

A. *Clinton Formation.*—The Clinton formation, in which the red ores occur, will be described somewhat fully here, owing to its particular interest in this connection. It consists in Alabama principally of sandstone, shale, and beds of impure hematite. The hematite-bearing beds, where unweathered, are calcareous and siliceous, ranging from a calcareous, richly ferruginous sandstone to a ferruginous, siliceous limestone. Beds of impure limestone are present in NE. Alabama, and in Georgia and Tennessee, but there appear to be no true limestone strata in the formation in the Birmingham district. The distinguishing feature of the formation is the relatively large quantity of iron oxide disseminated throughout all the sediments, either in the ferric or the ferrous state. While there are sharp lines of demarcation between certain beds of iron-ore, shale, and sandstone, many of these beds change from one to the other by gentle gradations. Consequently, there are in the section beds of ferruginous shaly sandstone and sandy shale, or the material may carry sufficient iron oxide to be styled a sandy ore or a shaly ore. As is well known, the ferruginous character of the Clinton formation is not peculiar to the southern Appalachians only, since rocks of equivalent age contain beds of hematite throughout the whole length of the Appalachians, as well as in such widely-separated localities as New Brunswick, New York, and Wisconsin.

In Alabama the formation is thickest toward the NE., measuring there more than 700 ft., and at this end of the area the proportion of shale is greatest. With thinning of the formation towards the S. and SW., where it becomes generally less than 300 ft. thick, and locally thins to less than 200 ft., the proportion of sandstone increases. It is difficult, however, to apply one set of rules to all the areas of the Clinton strata in Alabama, particularly to strips of the formations that lie on opposite sides of a valley and are separated by structures that are, in the main, anticlinal. At Birmingham, for instance, Red mountain and West Red mountain lie nearly parallel to each other, generally only 6 or 7 miles apart, yet there are greater differences in the character of the Clinton formation at corresponding points in these ridges than there are between points on the same ridge separated by two to three times that distance. Sections on West Red mountain cannot be correlated



Cp, Fort Payne Chert. *Sc*, Clinton Formation. *h*, Hickory Nut Seam. *i*, Ida

FIG. 1.—STRIKE-SECTION,

with those made 6 or 7 miles across the valley on Red mountain, but there is little difficulty in correlating sections made 12 to 15 miles apart on Red mountain. These conditions may perhaps be due to more abrupt changes in sedimentation from place to place in a direction at right angles to the shore-line of the body of water in which the sediments were deposited than parallel to this shore-line. There are, however, definite variations in composition from NE. to SW., as shown by a series of seven detailed sections on Red mountain, beginning NE. of Birmingham and continuing at irregular intervals along the strike of the beds for about 35 miles to the SW.,[3] as is illustrated in Fig. 1, a strike-section compiled from these detailed sections.

These sections showed that the thickness of the Clinton formation on Red mountain near Birmingham varied from less than 200 to 358 ft., that there were at least four distinct ore-horizons at certain places, with possibly a fifth horizon, represented by a very ferruginous sandstone locally developed, and that not all these ore-seams are persistent throughout the series of sections.

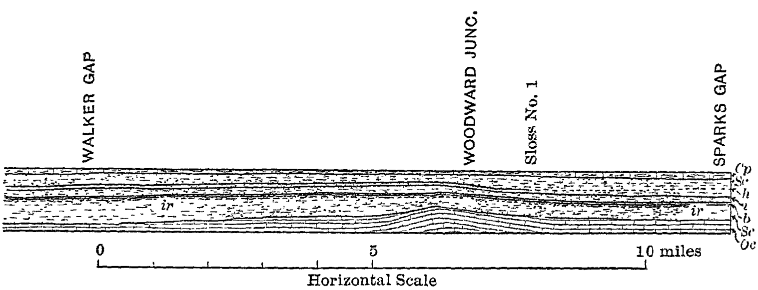
By way of contrast, six sections are given below, the first two being from NE. Alabama. Nos. 3 and 4 are typical of Red mountain in the Birmingham district, and Nos. 5 and 6 are characteristic of West Red mountain in the same district.

1. *Partial Section of Clinton Formation near Battelle, Ala.*[4]

	Thickness. Ft.	In.
Shale.		
Ore, dipping 23° to 45°, ...	4	6

[3] Burchard, E. F. The Clinton or Red Ores of the Birmingham District, Ala., *Bulletin of the U. S. Geological Survey* No. 315, pp. 136 to 139 (1907).

[4] Eckel, E. C. The Clinton or Red Ores of Northern Alabama, *Bulletin of the U. S. Geological Survey* No. 285, p. 174 (1906).



Seam. *b*, Big Seam. *Oc*, Chicamauga Limestone. *ir*, Irondale Seam.

CLINTON FORMATION.

	Thickness. Ft.	In.
Shale,	40	
Ore, dipping 23° to 45°,	2	
Shale,	300	
Ore, dipping 85°,	3	
Shale,	200	
Ore, dipping 85°,	3	4
Clinton strata exposed,	552	10

2. *Section of Well at Fort Payne Furnace.*[5]

	Thickness. Ft.	Depth. Ft.
Soil,	25	0 to 25
Chert (Fort Payne),	190	25 to 215
Black shale (Chattanooga),	12	215 to 227
Green and gray shale (Clinton),	340	227 to 567
Limestone, shale, and ore-seams,	18	567 to 585
Shale and ore-seams,	50	585 to 635
Shale, limestone, and sandstone,	180	635 to 815
Coarse ferruginous sandstone,	50	815 to 865
Sandstone and shale,	40	865 to 905
Limestone (Chickamauga),	.....	....
Total thickness of Clinton,	678	

3. *Section of Clinton Formation in Walker Gap of Red Mountain, near Birmingham.*

	Ft.	In.
Chert (Fort Payne).		
Shale, clay, and sand (Devonian),	1	6
Sandstone, massive,	16	10
Sandstone and shale,	13	7
Shale, drab to pink, with thin streaks of sandstone (partly concealed by débris),	85	4
Sandstone and shale alternating,	50	9

[5] McCalley, Henry. The Valley Regions of Alabama, pt. 2, *Alabama Geological Survey*, p. 154 (1897).

	Ft.	In.
Iron-ore (Big seam), top 8 to 15 ft. minable,	24	0
Sandstone and shale, with ore-seams in upper part,	13	7
Débris,	50	10
Shale, yellow, red and olive, with heavy sandstone interbedded,	41	0
Sandstone, heavy bedded,	3	5
Shale, yellow and red,	37	4
Base not exposed, but within distance of 20 ft.,	20	0
	358	2

4. *Section of Clinton Formation as Shown by Core from Diamond-Drill, at Sloss No. 1 Mine, on Red Mountain, near Bessemer.*

	Ft.	In.
Chert, solidly stratified (Fort Payne).		
Sandstone, red, with coarse grit,	5	8
Grit, coarse, soft, with gray sandstone,	5	8
Limestone, gray, hard, cherty,	6	7
Limestone (?), ferruginous,	31	0
Sandstone, ferruginous,	2	7
Sandstone, gray, extremely hard in places,	23	6
Sandstone, ferruginous,	40	7
Grit, very hard, fine, with reddish sandstone,	20	2
Iron-ore, limy (Hickory Nut seam ?),	2	7
Sandstone, gray,	7	11
Limestone (?), "marbleized,"	15	0
Sandstone, gray, hard,	1	11
Limestone, ferruginous (Ida seam ?),	5	8
Sandstone, ferruginous,	22	0
Iron-ore (Big seam), top 11 ft. minable		
Ore,	14	1
Sandstone, gray,	2	5
Shale, ferruginous,	...	8
Ore, limy,	2	5
Sandstone, highly ferruginous (Irondale seam ?),	4	8
Sandstone, mottled, highly ferruginous and fossiliferous (Irondale seam ?),	1	3
Calcareous rock, gray, with sandstone and shale interstratified,	30	0
	246	4
Bottom of formation probably within 35 ft.,	35	0
Probable total thickness of Clinton strata,	281	4

5. *Section of Clinton Formation on West Red Mountain, at Cunningham Gap.*

	Ft.	In.
Chert débris (Fort Payne).		
Sandstone, highly ferruginous, exposed in prospect-pit,	5	0
Concealed,	39	0
Sandstone,	7	10
Concealed,	31	6
Sandstone,	2	0
Concealed,	15	8
Sandstone, in massive beds,	31	6

	Ft.	In.
Concealed,	15	0
Sandstone,	4	0
Concealed,	23	9
Sandstone,	5	0
Concealed,	7	10
Sandstone,	1	0
Concealed,	15	7
Sandstone,	2	0
Concealed,	7	9
Sandstone, ferruginous (shown by prospect-pit),	2	0
Concealed,	15	8
Sandstone,	1	0
Concealed,	7	0
Sandstone, thick bedded,	23	8
Concealed,	31	6
Sandstone, thick bedded,	27	8
Shale,	2	0
Sandstone, thin to very thick beds, with shale partings,	102	0
Sandstone, thin bedded,	23	8
Concealed,	47	5
Sandstone,	4	0
Concealed,	5	0
Limestone, impure and ferruginous (probably top of Chickamauga),	...	...
	507	0

6. *Section of Upper Part of Clinton Formation in Gap of West Red Mountain, near Dale, Ala.*

	Ft.	In.
Shale, black (Devonian),		
Sandstone, green predominating, gray and red, evenly bedded, with shale partings, most numerous at top,	110	0
Shale, yellow-green,	5	6
Sandstone, gray, thin bedded,	10	6
Iron-ore,	2	0
Concealed,	6	0
Sandstone, ferruginous, with decomposed ore,	6	0
Iron-ore, lean, limy, fossiliferous,	10	0
Concealed by very red soil and red sandstone débris,	210	0
Limestone (Chickamauga),	...	...
	360	0

Workable ore occurs in one bed near Dale, sections of which are given on p. 107.

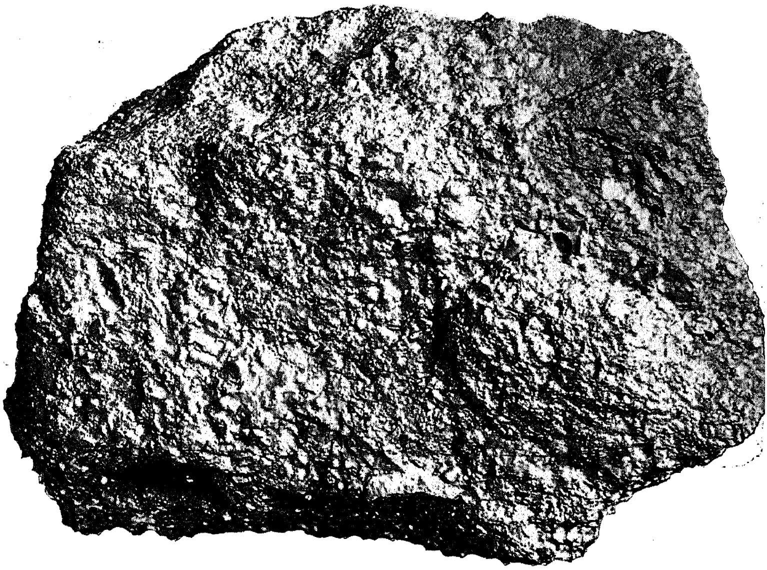
The foregoing sections show, besides the variation in details and thickness of the formation, the relation of the ore-beds in each district. On Red mountain in the Birmingham district four beds have been recognized and named by the miners. These beds are all shown in section 4, but in section 3 only one seam

is clearly shown. Further details of these ore-beds will be given later in this paper.

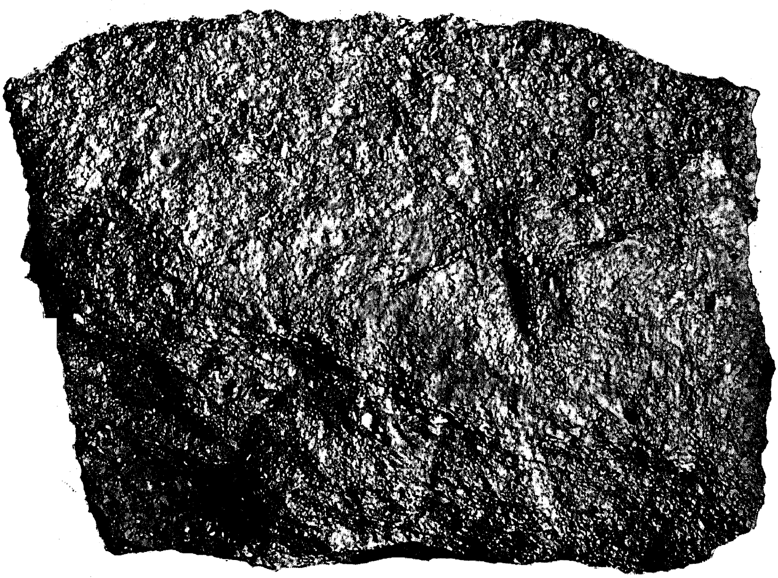
The Clinton formation in West Red mountain is evidently slightly thicker than in Red mountain and there are differences in the composition of the beds and in the character of the ore-seams that are difficult to account for in the short distance of 5 or 6 miles that separates the two limbs of the anticline, especially since along the strike the rocks are fairly constant in character for two or three times that distance. Possibly the sediments that constitute these two ridges were not deposited as one continuous sheet, and therefore the strata may never have been connected, as has heretofore been supposed. Such conditions might be accounted for in part by the existence of strips of land or barriers within the Clinton sea.

2. *Structure.*

The geologic structure of the valley-regions in which the Clinton rocks are exposed is in general anticlinal, with comparatively shallow synclines inclosed between the major anticlines. The anticlines are relatively long and narrow; their general trend is N. 30° to N. 40° E., and they are usually steeper on their NW. limbs, or are overturned, and in places broken by overthrust faults. Faults occur also within the anticlines, resulting in places in the repetition or duplication of strata. Besides these systems of strike-faults there are a number of minor cross-faults which are probably normal, and which offset the outcrop of the formations which they cut. The rocks brought to the surface by these anticlinal folds are for the most part pre-Pennsylvanian, and the Pennsylvanian or "Coal Measures" flanking the valleys flatten out to the NE. and SE., and constitute the several coal-fields of the State. On the borders of the valleys where the structural relations are normal the Clinton strata are found dipping away from the anticline, and passing below Devonian and Mississippian rocks with dips that vary from 10° to vertical, and where the rocks have been badly disturbed the Clinton formation is found with reversed dips due to overturning, or it may be faulted completely out of the section. Complicated structure has rendered unworkable a considerable quantity of good ore, although it is probable that the choicest ores, considered from

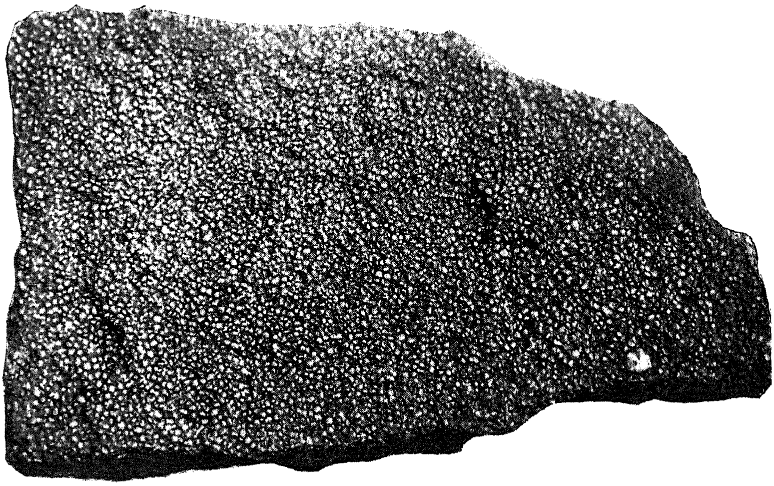


Mixed Oölitic and Fossil Ore.

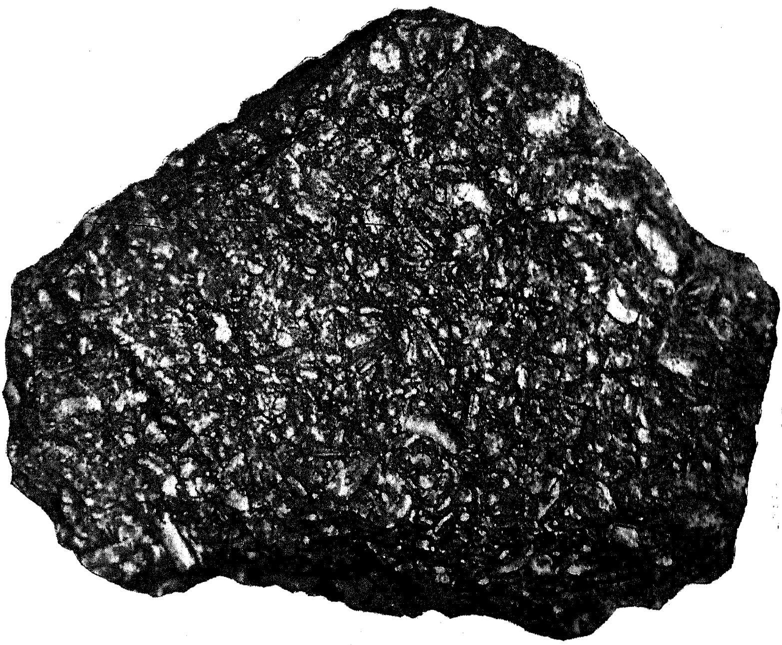


Fossil Ore.

FIG. 2.—CLINTON IRON-ORE FROM THE BIRMINGHAM DISTRICT, ALABAMA.



Oölitic Ore.



Fossil Ore.

FIG. 3.—CLINTON IRON-ORE FROM CLINTON, N. Y.

the stand-points of both quantity and quality, have been the least disturbed.

3. *The Ore.*

A. *Character.*—The Clinton iron-ore is of a sedimentary, bedded type. In Alabama, as in the Clinton locality in New York, the ore is of two varieties, viz., the "oölitic" or "flaxseed" ore, and the "fossil" ore. Typical specimens of the Alabama varieties are shown in Fig. 2, and of the New York varieties in Fig. 3. Both of these two varieties of ore, where they have not been leached by meteoric waters or subjected to the agencies of weathering, are found in the form known as "hard" ore, and both, as a result of leaching, become altered to "soft" ore. The oölitic ore consists of compact grains or concretions having concentric shells of hematite and films of argillaceous material with a core of silica. The concretions are cemented together by ferruginous calcium carbonate. The silica nuclei are rounded, and in places most of them are of a lenticular or flaxseed shape, and where they are of this shape they are sometimes found to run into somewhat coarse grains, or fine pebbles. Much of the ore of the Big seam, opposite Birmingham, is of this character.

The fossil ore consists of fragments of fossils such as crinoids, bryozoans, and brachiopods. The composition of the fossil fragments varies from nearly pure calcium carbonate to ferric oxide mixed with little or no lime, the mass being cemented together by more or less ferruginous calcium carbonate or ferric oxide. The fragments are usually rather finely comminuted, but occasionally a fairly well-preserved fossil is found.

In some places ore-beds have been observed to be composed wholly of either flaxseed ore or fossil ore, but in many places the thicker beds contain seams of one variety interbedded with a mass of the other variety of ore, and even a mixture of fossil and granular ore has been found. Shale partings are common, especially near the top and bottom of the ore-beds.

According as the ore is high or low in lime it is termed "hard" or "soft" ore. The distinction between the two varieties is based on differences in their chemical composition rather than on differences in hardness, although the terms "hard" and "soft" as originally applied to the ores probably had reference to their physical condition, since on the outcrop the soft

ore is in general rather porous and friable. The unaltered ore is of the hard variety. The soft ore has resulted from the leaching by percolating waters of the soluble calcium carbonate contained in the hard ore. This alteration occurs at the outcrop of the ore-beds and down the dip to varying distances, depending on the thickness and permeability of the cover. Where the strata dip at fairly high angles and are underlain by impervious shale, the conditions are favorable for the passage of water through the beds to considerable depths. In a few places pockets of soft ore, surrounded by unleached ore, are encountered at relatively long distances from the outcrop. Such an occurrence of soft ore is usually due to the presence of fissures or brecciated rock, through which surface-water has reached the ore-bed. Where the overlying cover is heavy at the mouth of a slope or tunnel, the soft ore rarely extends more than 50 ft., but here and there the ore has been well leached to a distance of 400 ft. from the outcrop. With the removal of the calcium carbonate from the original ore the relative percentages of the remaining less-soluble constituents, mainly iron oxide, silica, and alumina, are increased. The analyses in Table II. show at the left a typical hard ore and at the right a typical soft ore, with semi-hard grades between.

TABLE II.—*Analyses of Clinton Iron-Ores, Showing Gradation from Hard to Soft Ore.*

	1.	2.	3.	4.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Iron, metallic (Fe),	37.00	45.70	50.44	54.70
Silica (SiO_2),	7.14	12.76	12.10	13.70
Alumina (Al_2O_3),	3.81	4.74	6.06	5.66
Lime (CaO),	19.20	8.70	4.65	0.50
Manganese (Mn),	0.23	0.19	0.21	0.23
Sulphur (S),	0.08	0.08	0.07	0.08
Phosphorus (P),	0.30	0.49	0.46	0.10

The analyses in Table II. represent ore-samples from a single slope on the same horizon of the Big seam in Red mountain, near Birmingham, at distances respectively of 540, 480, 420, and 240 ft. from the mouth of the slope. Beyond the point at which No. 1 occurs there is no great change in the character of the ore, for, as mined at present, the bed carries an average of 35 per cent. of metallic iron in this particular mine.

Although the soft ore carries a higher percentage of iron, the hard ore has the advantage of containing almost or, in places, quite enough lime to flux the silica that it contains. In

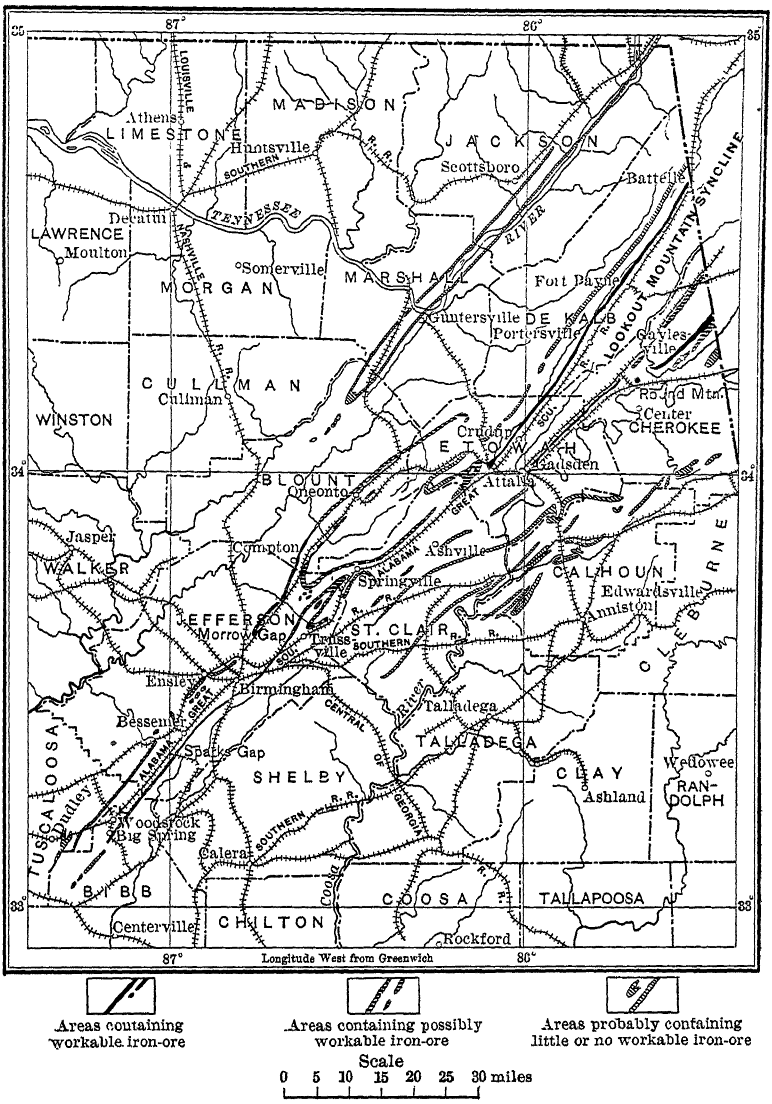


FIG. 4.—SKETCH-MAP OF NORTHEAST ALABAMA, SHOWING OUTCROPS OF CLINTON FORMATION.

case a hard ore contains more lime than is needed to flux its silica, soft ore or brown ore (limonite) may be added to the burden to take up the excess of lime.

The ore in Alabama occurs in well-defined beds, one to four of which may be found in nearly any complete section of the Clinton formation. Details as to certain of these ore-beds will be given in the description of the Birmingham district.

B. *Geographic Distribution.*—The Clinton iron-ore outcrops in the marginal ridges of most of the long narrow valleys in E. and NE. Alabama. Fig. 4 is a sketch-map of NE. Alabama, showing the outcrops of the Clinton formation. These valleys trend NE–SW., parallel to the Appalachian axis, and, as is noted in the paragraph on Structure, the valleys are for the most part developed along the axes of anticlinal folds, and the strata containing the iron-ore beds dip away from the valleys and pass below later rocks that form the ridges and plateaus of the coal-fields. Since the width of outcrop is nearly everywhere narrow, ranging from the thickness of the formation where it dips steeply, to 2,000 ft. or more where the beds dip gently, the areal outcrop of the Clinton is comparatively small. The linear extent, however, of the many strips of the formation that have been brought to the surface by folding and erosion totals nearly 700 miles, although there are probably hardly 100 miles of outcrops that contain workable beds of ore.

There are two localities in Alabama in which the Clinton ore has been worked on an important scale, viz., along Birmingham valley from above Birmingham to below Bessemer, and along Little Wills valley between Attalla and the Georgia line. This last-named locality is on the western border of the Lookout Mountain syncline. Ore has been mined also at several places outside of the two localities mentioned, as, for instance, near Dudley, about 35 miles SW. of Birmingham, and at Gaylesville, Round mountain, and other points east of the Lookout Mountain syncline.

III. Local Descriptions.

1. *Birmingham District.*

A. Topography and Its Relations to Industrial Development.—By the Birmingham district is meant the area from which the blast-furnaces at Birmingham, Ensley, and Bessemer derive their iron-ores. It lies within the valley-region of north-central Alabama. The valley topography is characterized by long, narrow, canoe-shaped troughs, in general parallel to each other

and separated by well-defined ridges. The trend of the troughs is approximately N. 30° E. Their form is directly dependent on the geologic structure and lithology of the underlying rocks. They are developed mainly on the softest and most soluble rocks, along the axes of anticlines, the most enduring strata on the limbs of the folds forming the rims of the valleys. At distances of 2 to 5 miles apart openings or "gaps," some of which extend to the valley-level, are cut at right angles through the ridges and afford convenient passageways between the valleys.

Birmingham valley, the largest and from an industrial standpoint the most important of these valleys, extends NE–SW. from the vicinity of Springville to beyond Vance, and NW–SE. from the Warrior coal-field, or Sand mountain, to the Cahaba coal-field, or Shades mountain. Birmingham valley is divided into minor valleys by low ridges, such as Red mountain and West Red mountain, Flint ridge, and Cemetery ridge, all due to folding and faulting of the main anticline; and this complicated structure finds expression in Shades valley, S.E., Roups valley, SW., Jones valley, NE., and Opossum valley, NW.

Red mountain, the main minor ridge within Birmingham valley, yields nearly all the red ore of the district, and the Woodstock area, in the SW. portion of the valley, produces the major part of the brown ore. Coking-coal is mined in the Warrior coal-field, only a few miles distant from the blast-furnaces. Dolomite, at the base of the Knox formation, and limestone, of Ordovician and Mississippian age, suitable for fluxing, occur in the valley, stratigraphically below and above the red ore, as shown in Fig. 5.

The simple, regular topographic features of the valley, in connection with the structure and distribution of the rocks, have made accessible the ores and limestone at nearly every point where they are of workable character, and enterprising railroad and mining companies have rapidly improved the opportunities for developing the region.

B. *Distribution and Structure of Formations.*—A generalized section NW. from the Cahaba coal-field across the valley at Birmingham would expose the rocks in the order of Table I., reading from the top downwards, with varying dips on the SE.

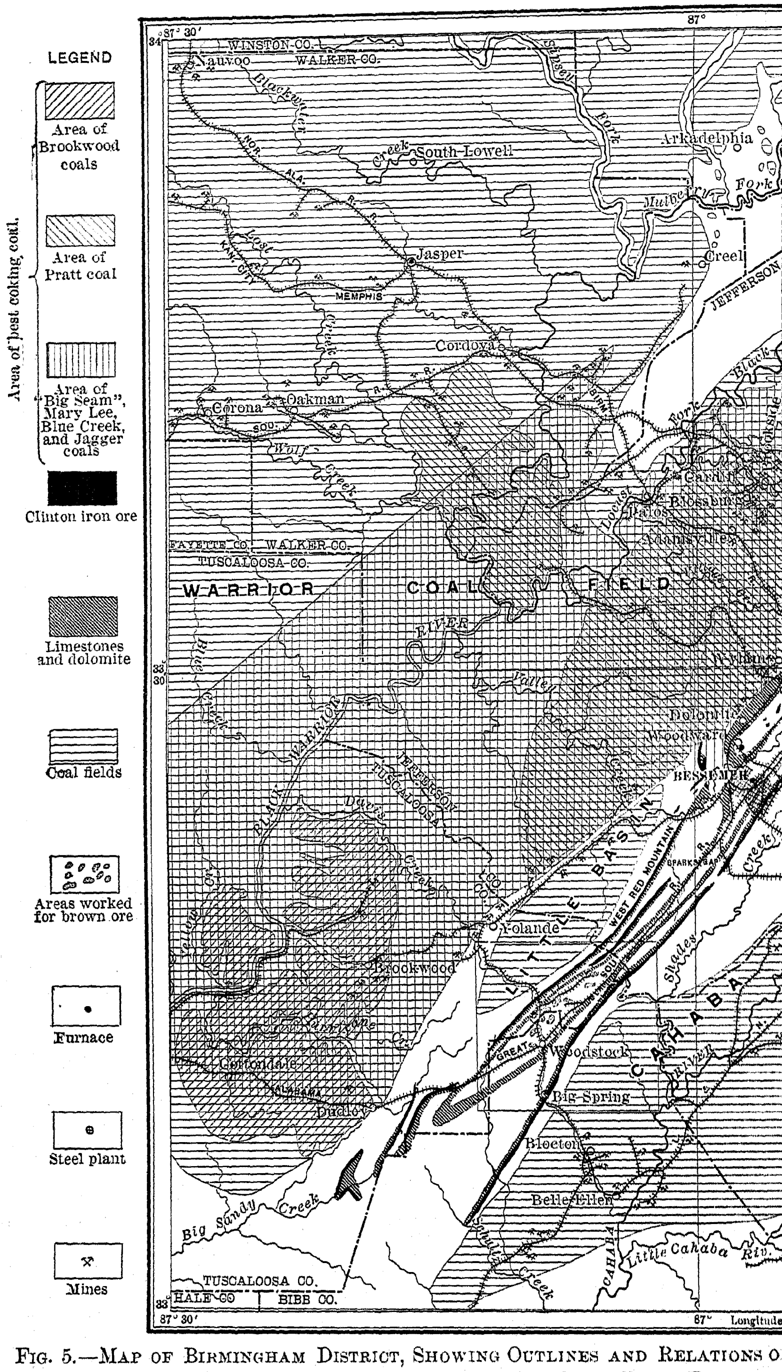
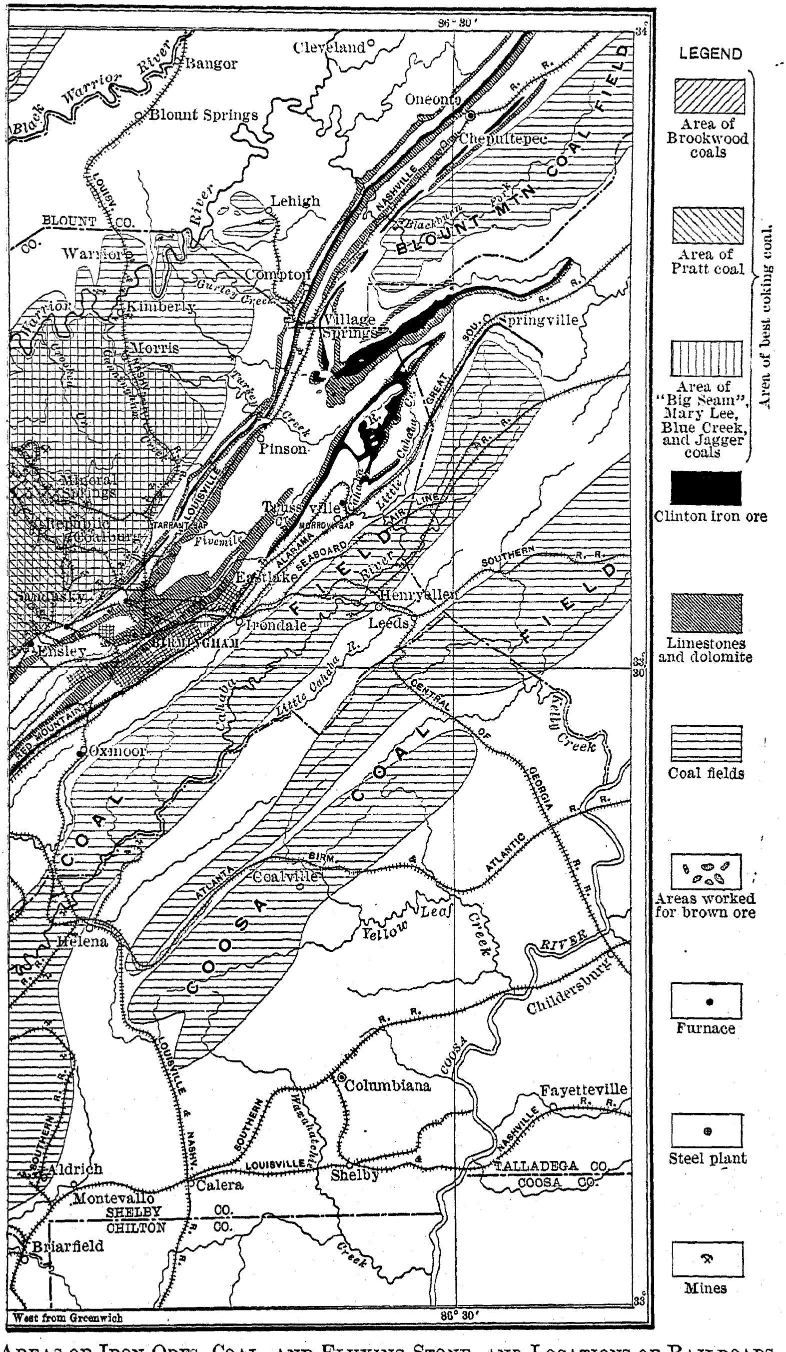


FIG. 5.—MAP OF BIRMINGHAM DISTRICT, SHOWING OUTLINES AND RELATIONS OF MINES, BLAST-FURNACES,



AREAS OF IRON-ORES, COAL, AND FLUXING-STONE, AND LOCATIONS OF RAILROADS, AND STEEL-PLANTS.

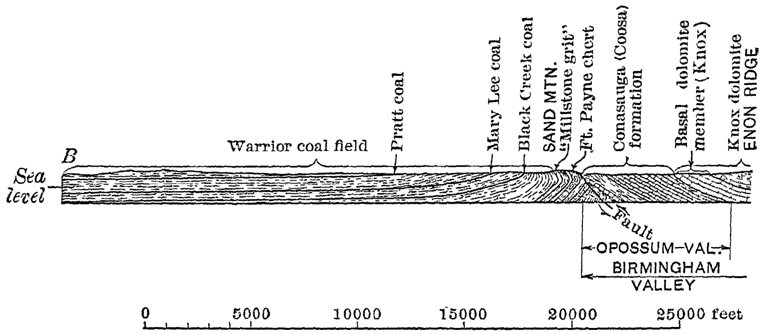
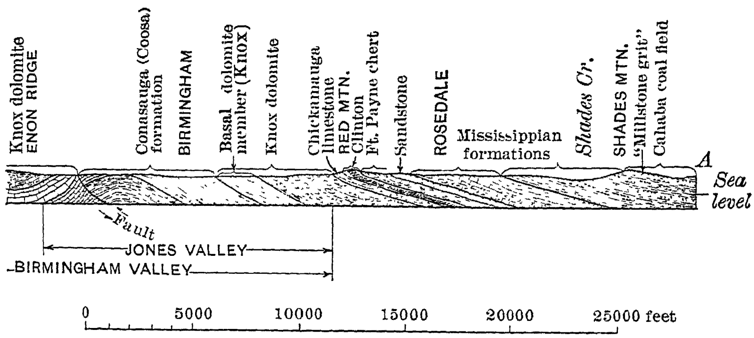


FIG. 6.—GEOLOGIC-STRUCTURE SECTION ACROSS VALLEY

flank of a non-symmetrical anticline having the Conasauga shale and limestone on the axis of the fold, as shown in Fig. 6. A minor syncline follows, faulted down to the SE., with the Knox dolomite held in the basin. The Conasauga again appears to the NW. of the Knox syncline, and bordering the Conasauga, on the NW. side of the valley is an overthrust fault, which brings the basal sandstone of the Coal Measures in contact with Cambrian and Ordovician rocks, and overturns the rocks adjacent to the fault. The Clinton sandstone, dipping SE., forms the crest of Red mountain, with the Fort Payne chert overlying it, and the Chickamauga limestone underlying it, making, respectively, the SE. and NW. slopes of the mountain. Where the throw of the overthrust fault has not been great enough to bury the Silurian rocks completely below the surface, the Clinton formation, dipping steeply, is exposed in a narrow outcrop. The presence of this formation in places on the NW. side of the valley has given to the ridge the name West Red mountain. All the formations may be considered to extend longitudinally throughout the valley, in practically the relationships indicated. The minor folding and faulting have duplicated the strata of West Red mountain in the southern part of the valley, forming McAshan mountain and a low ridge partly buried by post-Palæozoic sediments, south of Dudley. At the NE. end of the valley the extent of outcrop of the Clinton and associated strata is increased by the synclinal Blount mountain with the outlying Cedar mountains, and by the synclinal structure north of Trussville.

Red ore occurs in practically all the outcrop-areas of the



AT BIRMINGHAM, ALA. (BY CHARLES BUTTS.)

Clinton formation, but only in Red mountain has it been found of sufficient thickness and purity to be worked on an important scale. The availability of the ore depends largely on the attitude of the inclosing strata. The beds of Red mountain dip SE. at moderate angles, 10° to 30°, which are in the main fairly constant for a mile or more along the strike, and for a quarter to a half a mile down the dip. Locally, there are abrupt "rolls" or changes in the dip due to minor folds parallel to the main axis, and in some places the ore has been so faulted that efforts to find it or to exploit it further have been suspended.

The questions of greatest importance to the district are (1) whether the present quality of the ore will be maintained to great depth; (2) whether the present workable thickness will continue to whatever depth it may be possible to mine the ore; and (3) whether the structure of the rocks underlying Shades valley—below which the ore-beds normally lie SE. of Red mountain—will be favorable for mining the ore. Certain facts have been brought out during recent surveys which have a bearing on these questions.

The areal mapping by Charles Butts[6] has indicated that the structure in Shades valley is not regular. Mr. Butts finds that the dip of the rocks of Red mountain and of those on both sides of Shades valley, as well as of those at many points in the valley itself, appears to average about 15° to the SE. Since, however, a dip of 15° would give a depth of 1,500 ft. to the

[6] Map to accompany *Bulletin of the U. S. Geological Survey* No. —, The Iron-Ores, Fuels, and Fluxes of the Birmingham District, Ala. (in preparation).

Clinton formation, and a thickness of 1,200 ft. for the Mississippian limestone and shale on the east side of Shades valley, this apparent average dip probably does not continue, because near the east side of Shades valley Clinton ore is reported at 900 ft. Considerable structural irregularity is also indicated by other field-observations, although on account of lack of exposures only a meager knowledge of the structure has been obtained. On Shades creek, 8 miles south of Bessemer, NW. dips of from 20° to 25° were observed, and showed the presence of at least two folds, which interrupt the general SE. dip. One of these anticlines undoubtedly extends up Shades creek to the Southern railroad, where a bore-hole reached a bed of red ore at less than 300 ft. below the surface. A considerable expanse of coarse-grained light-gray limestone lies along Shades creek in this region, which resembles a limestone noted north of Trussville between the Fort Payne chert and the overlying sandstone, and if it is such it necessarily indicates an anticline by which it is brought to the surface. Exposures are fairly good in Shades valley from the latitude of Readers gap southward, and the rocks can be seen to lie nearly flat over wide areas, a fact that is indicated by the wide area of Mississippian shale in that region. Just north of Readers gap there are evidences of minute puckerings of the strata. A short distance south of Hall's spring the rocks are folded into a small U-shaped syncline, and near by is a narrow outcrop of a vertical rock. On the Louisville & Nashville railroad, about half a mile south of Graces station, a sharp overturned fold is exposed in a cut. At a number of points very steep SE. dips were observed. Besides the facts stated above, a number of faults, of greater or less magnitude, and a good many flexures have been encountered in the various ore-mines of the region. The facts cited indicate that the structure of Shades valley is irregular, though it is impossible, owing to the general concealment of the strata, to give any detailed account of the number and magnitude of the irregularities. There may be some small irregularities or a few great ones, or there may be both, the last being the most probable supposition.

In planning future operations beneath Shades valley, the importance of taking into consideration the certainty of some structural irregularities, such as sharp folds, and faults, and the

possibility of unknown and perhaps greater disturbances, and their effect upon the expense of ore-mining, will be apparent from the foregoing statements. Careful prospecting with a core-drill is recommended as probably the only method that can be relied upon to yield definite information as to the ore-reserves below this valley.

C. *Divisions of the Birmingham District.*—Owing to the considerable extent of Birmingham valley, to the distribution of the ore-beds along the margins and at the ends of the valley, and to the variation in the character of the ore from place to place, the district is divided for convenience of description and estimates of ore-reserves into seven parts, A to G inclusive. The basis for division is mainly geographic, and the order of divisions from A to G represents in a general way the commercial importance of the divisions, based on (1) quality of ore; (2) quantity of ore; (3) structure of ore-beds; (4) accessibility; and (5) distance from blast-furnaces. It should be understood, however, that this outline of divisions is not intended as a definite estimation or appraisal of relative values. The mass of data obtained by recent field-work in the district will be presented in a bulletin of the Geological Survey soon to be published.[7]

Division A.—Division A includes that part of Red mountain extending from Morrow gap SW. to Sparks gap, a distance of about 26 miles. All but two of the productive mines of the district are in this strip of Red mountain. The total number of workings, including slopes, open-cuts, and combination mines, in active operation in 1906, was thirty. These mines are served by the Birmingham Mineral Division of the Louisville & Nashville railroad, which is built along the slope of the mountain from 100 to 350 ft. below the summit of the ridge. The road runs first on one side of the mountain, then on the other, threading its way back and forth through several natural passageways, such as Sadlers gap, Lone Pine gap, Walker gap, and Readers gap. From the latter gap the road leads to Bessemer, with a spur extending SW. along the ridge to the Potter slopes. Some mines, especially those where the road passes along the west

---

[7] *Bulletin of the U. S. Geological Survey* No. —, The Iron-Ores, Fuels, and Fluxes of the Birmingham District, Ala. (in preparation).

side of the mountain, are so situated that their tipples can be built directly on a siding of the railroad. Others, facing the east, have built spurs reaching back into the lateral ravines. Through Red gap, between Irondale and Gate City, five railroads enter Birmingham from the east and north. At Graces gap the Louisville & Nashville railroad passes southward across the Cahaba coal-field to the Gulf, and at Sparks gap the Southern railway finds an outlet southeastward. Therefore this portion of the district is well supplied with transportation-lines, and consequently its development has been facilitated. The numerous open-cuts and slopes afford excellent opportunities for study of the ore-beds, whose description follows.

a. Description of the Ore-Beds.—In section 4 of the Clinton formation, on p. 80, the relation of the important ore-beds on Red mountain is shown, as also in Fig. 1. The following summary gives the important facts regarding these beds:

Hickory Nut Seam.—This bed consists of from 3 to 5 ft. of siliceous ore or ferruginous sandstone, characterized by an abundance of *Pentamerus oblongus*, fossils which resemble hickory nuts incased in the partly open outer shucks. The ore is of too low a grade to be worked at present. The bed is thickest between Birmingham and Bessemer, and where recognized it lies about 12 to 20 ft. above the next lower ore.

Ida Seam.—This bed consists of from 2 to 6 ft. of rather siliceous ore, associated with from 14 to 16 ft. of ferruginous sandstone. Ore at this horizon is more continuous and extensive than at the horizon of the Hickory Nut seam. It has been recognized at many of the workings from Bald Eagle gap to beyond Clear Branch gap. Soft ore, from 3 to 5 ft. thick, has been obtained from it in a few surface-workings. Such ore carries from 35 to 44 per cent. of metallic iron, with a corresponding range in silica of from 45 to 32 per cent. The Ida generally occurs from 20 to 50 ft. above the top of the Big seam.

Big and Irondale Seams.—These two ore-beds are considered together, since they are generally rather closely associated in space. The ore, however, is somewhat different in quality, and the beds are so sharply separated by sandstone or shale that they may be mined independently.

The thickness of the Big seam is variously estimated at

from 16 to 30 ft. It extends as a traceable unit on Red mountain practically the whole length of the mining-district. Notwithstanding the great thickness, there is rarely more than from 10 to 12 ft. of good ore in a single bench, and at most places only from 7 to 10 ft. is mined. Probably the maximum thickness of the whole bed is attained between Red gap (near Irondale) and Bald Eagle, although for a mile SW. of Red gap it remains nearly as thick. From NE. to SW. the total thickness of the ore-bearing strata gradually decreases, without, however, altering greatly the thickness of the workable portion. About the middle of the district the bed becomes separated into two benches, either by a well-defined parting along the bedding-plane or by a shale-bed, thin at first, but thickening gradually to the SW. The middle of the Big seam is the workable part in the NE. end of the district, but the upper bench is of the most importance throughout the rest of the area. In the SW. portion of the district the lower bench, which farther NE. is composed of ore that will in later years be considered good enough to mine, becomes a series of thin strata of lean ore and shale, and is consequently of no possible value; and finally, the upper bench itself becomes shaly and carries only a very low-grade ore.

The best part of the Irondale seam is on Red mountain between Pilot knob on the NE. and Lone Pine gap on the SE. Southwest of Lone Pine gap the bed either consists of interbedded low-grade iron-ore and shale, or else its identity is completely lost. Its soft ore, now nearly all mined out either by surface-trenches or slopes, is the best in the district. Its hard ore is also of high grade, and hitherto has been, for the most part, held in reserve, since ore could be produced from the thicker Big seam at a lower cost per unit of iron.

The structure and composition of the Big and Irondale seams are shown in the following series of sections, taken at intervals of 2 to 5 miles apart along Red mountain, beginning in the NE. portion of the mining-district:

Character of Big and Irondale Seams 1 Mile Northeast of Red Gap, near Irondale.

Strata.	Thickness.		Character.
Sandstone.			
Big seam:	Ft.	In.	
Ore, sandy, . . .	1	8	Metallic iron, 16 to 20 per cent. ; insoluble, 40 ± per cent. ; lime, 18 ± per cent.
Ore, lean, with fine quartz-pebbles, .	5	0	
Ore, massive, cross-bedded, mined, .	7	0	Hard ore, averages metallic iron, 36 per cent. ; insoluble, 26 per cent. ; lime, 20 per cent.
Ore, similar in appearance to above, but not mined at present, . . .	6	0	Percentage of iron grades down from 35 at top to less than 20 at bottom ; insoluble rises to more than 60 per cent.
Sandstone, ferruginous, lean ore, and shale, ...	20	0	
Shale, ...	0 to 6	...	
Sandstone, very hard,	3	0	
"Gouge," calcareous,	...	6	
Irondale seam:			
Ore, mined, . . .	5	0	Semi-hard ore, averages metallic iron, 37 per cent. ; insoluble, 29 per cent. ; lime carbonate, 14.25 per cent.
Shale, hard.			

Character of Big and Irondale Seams Half a Mile South of Red Gap.

Strata.	Thickness.		Character.
Sandstone, coarse, ferruginous.			
Big seam:	Ft.	In.	
Ore, not mined, contains much silica in coarse grains and small pebbles, . . .	22	0	Upper half, soft ore : Metallic iron, 22 ± per cent. ; insoluble, 64 ± per cent. ; lime, trace. Lower half, soft ore : Metallic iron, 32 ± per cent. ; insoluble, 47 ± per cent. ; lime, trace.
Shale, ...	...	1	
Ore, mined, ...	7	0	Soft ore : Metallic iron, 36 ± per cent. ; insoluble, 45 ± per cent. ; lime, trace.
Ore, not mined, ...	10	0	Semi-hard ore : Metallic iron, 25 ± per cent. ; insoluble, 50 ± per cent. ; lime carbonate, 8.12 per cent.
Shale, soft, ...	3	0	
Irondale seam:			
Ore, mined, ...	4	4	Soft ore : Metallic iron, 50 ± per cent. ; insoluble, 15 ± per cent. ; lime, trace.

The two following sections, made within the next 5 miles to the SW., show that although the total thickness of the iron-bearing strata in this direction grows gradually less, yet the thickness of workable material remains fairly constant.

Character of Big and Irondale Seams near Lake View, Birmingham.

Strata.	Thickness. Ft.	In.	Character.
Sandstone, thin bedded.			
Big Seam:			
Ore, mined, ...	10	0	Soft ore: Metallic iron, 40 ± per cent.; insoluble, 39 ± per cent.; lime, trace. Hard ore: Metallic iron, 34 ± per cent.; insoluble, 26 ± per cent.; lime, 20 ± per cent.
Shale, ...	...	8	
Ore, not mined, . . .	7	0	Value decreases regularly downward. Soft ore: Metallic iron, 15 to 25 per cent.; insoluble, 50 to 60 per cent.
Shale, ...	2	0	
Irondale seam:			
Ore, ...	2	8	Hard ore: Metallic iron, 38 ± per cent.; insoluble, 16 ± per cent.; lime carbonate, 24 ± per cent. Soft ore: Metallic iron, 47 ± per cent.; insoluble, 26 ± per cent. Only hard ore mined at present.
Shale, ...	...	9	
Ore, ...	2	2	

Character of Big and Irondale Seams near Lone Pine Gap.

Strata.	Thickness.		Character.
Shale.			
Big seam:			
Ore, mined, . . .	10 ±	0	Hard ore: Metallic iron, 36 ± per cent.; insoluble, 25 ± per cent.; lime carbonate, 20 ± per cent. Soft ore: Metallic iron, 44 ± per cent.; insoluble, 35 ± per cent. Semi-hard ore mined at present.
Shale, ...	1 +	...	
Ore, not mined, . .	6	6	Deteriorates in value regularly downward, top ore being poorer than the ore mined above.
Shale, ...	2	0	
Irondale seam:			
Ore, shaly, not mined,	6	0	Low-grade ore interbedded with shale.

The following section illustrates the complete deterioration of the Irondale seam:

Character of Big and Irondale Seams at Open Cut, Greenspring Mine.

Strata.	Thickness. Ft.	In.	Character.
Sandstone, coarse, ferruginous.			
Shale, yellow.			
Big seam:			
Ore, massive, cross-bedded, and jointed; mined,	8	0	Soft ore: Metallic iron, 42 ± per cent.; insoluble, 31 ± per cent.; lime, 2 ± per cent. Semi-hard ore: Metallic iron, 38 ± per cent.; insoluble, 32 ± per cent.; lime, 8 ± per cent. Mostly semi-hard ore mined at present.
Parting on bedding-plane.			
Ore, rather a ferruginous sandstone or coarse grit; mined in only a few places,	8	0	Lower bench.
Shale,	...	2	
Sandstone, ferruginous and shaly,	1	0	
Shale, sandy,	...	6	
Sandstone, ferruginous,	1	3	
Shale,	...	2	
Sandstone,	...	4	
Shale,	...	2	
Ore, sandy,	...	5	
Shale,	...	5	
Ore, sandy,	...	3	
Shale,	...	2	
Irondale (?) seam:			
Ore, very sandy,	1	6	Not minable.
Shale,	...	2	
Sandstone, fine-grained, very ferruginous,	1	4	
Shale,	...	2	
Sandstone, fine-grained, very ferruginous,	...	10	
Shale,	...	1	
Sandstone, very ferruginous,	...	5	
Shale,	...	1	
Sandstone, very ferruginous,	...	10	
Shale.			

At Graces gap, 1.5 miles farther SW., only the Big seam appears to be present. It has a thickness of about 22 ft. here,

and the upper bench, from 10 to 12 ft. of ore, is mined. Two miles SW. of Graces gap the ore presents the following phase:

Character of Big and Irondale Seams at Mouth of Slope No. 12, Tennessee Coal, Iron & Railroad Co.

Strata.	Thickness. Ft.	In.	Character.
Shale and sandstone in thin beds.			
Big seam:			
Ore, mined, . . .	8 to 10	0	Hard ore: Metallic iron, 35 ± per cent.; insoluble, 18 ± per cent.; lime, 16 ± per cent. Only hard ore mined at present.
Shale, thin parting.			
Ore, lean and siliceous, with a few local shale partings, ...	9	0	Not minable under present conditions.
Ore, oölitic and fossiliferous, in thin bands alternating with streaks of calcite and shale, .	2	1	
Ore, shaly, ...	1	3	
Ore, ...	...	4	
Shale, ...	...	8	
Irondale (?) seam:			
Ore, siliceous, . .	...	6	Not minable.
Shale, ...	...	1	
Ore, siliceous, . .	...	8	
Shale, ...	...	3	
Ore, very sandy, .	1	3	
Shale, ...	...	1	
Sandstone, ferruginous, ...	...	7	
Shale, sandy.			

Five miles SW. of slope No. 12, along Red mountain opposite Bessemer, the parting between the upper and lower benches of the Big seam reaches a thickness of 3 ft. in places. The upper bench maintains its usual quality and its thickness of from 10 to 11 ft., from 8 to 10 ft. of which are taken in mining. Thin streaks and lenses of shale begin to appear near the top and bottom of this bench. The lower bench of the Big seam has dwindled down to 4 or 5 ft. in thickness and is generally composed of alternating thin strata of ore and shale.

The Irondale seam evidently has not been recognized

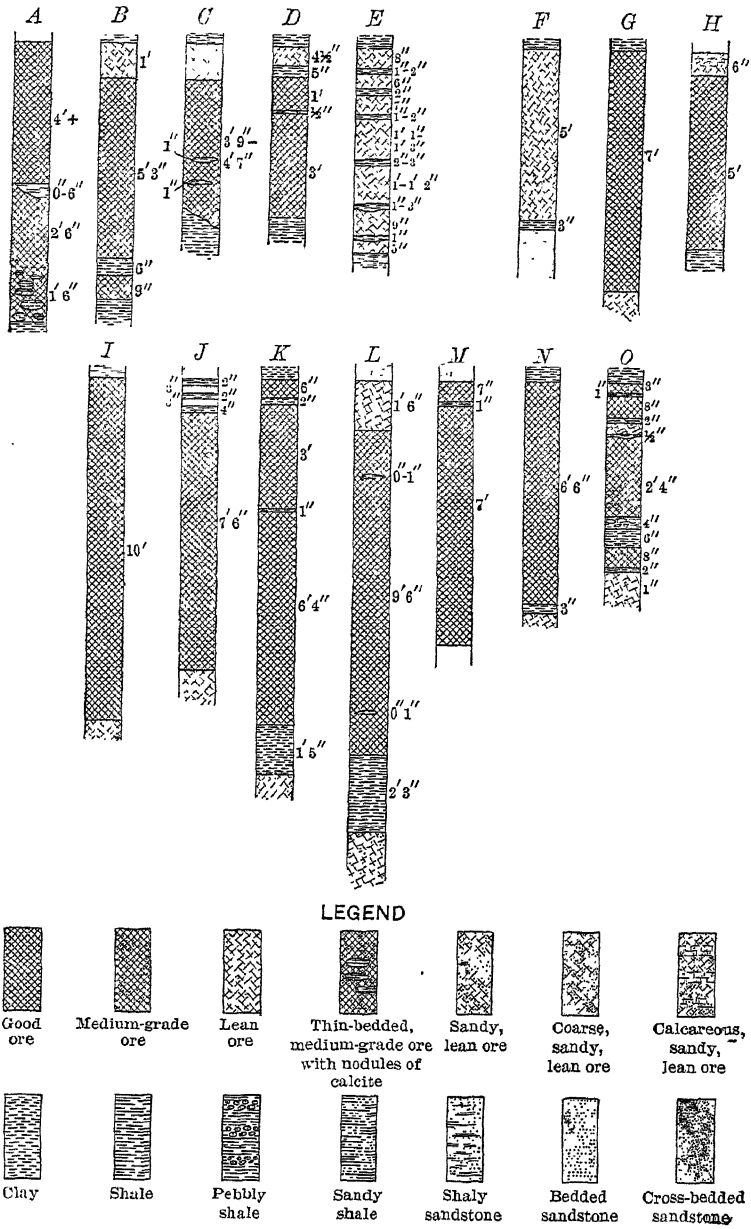
here. Three miles farther SW. the seams show the following section:

Character of Big and Irondale (?) Seams at Mouth of Potter Slope No. 1, Tennessee Coal, Iron & Railroad Co.

Strata.	Thickness. Ft.	In.	Character.
Shale.			
Big seam:			
Ore, solid, mined, . .	8	0	Soft ore: Metallic iron, 47 ± per cent.; insoluble, 24 ± per cent.; lime. 0.80 ± per cent. Soft ore mined at present.
Shale, ...	1	6	
Sandstone, shaly, ferruginous, ...	...	9	Lower bench.
Ore, ...	...	3	Lower bench.
Shale, ...	...	1	Lower bench.
Sandstone with shaly partings, ...	1	6	Lower bench.
Shale, ...	3	0	Lower bench.
Horizon of Irondale seam (?):			
Ore, sandy, lean, with shale-partings, . . .	1	0	

Ore-sections of the deposits at various mines in Alabama are shown diagrammatically in Fig. 7.

In July, 1906, the deepest slope in Red mountain was reported to be more than 1,800 ft. long. Three other slopes had been driven nearly 1,800 ft. each, and there were twelve slopes between 900 and 1,500 ft. long. All the slopes 900 ft. or more in length are in the strip of Red mountain below Birmingham. The newer mines at the extremities of the district have slopes ranging between 200 and 800 ft. in length. The deepest slope goes down on beds whose average dip is about 22°, so that its present depth is about 650 ft. below the level of the valley at a point directly above the bottom of the slope. Projected at the same angle to a point directly below Little Shades creek the slope would have a length of about 6,400 ft. and a depth below the creek of 1,800 ft. It is not known whether the ore extends with an unchanged dip and thickness to this depth. Drill-records obtained farther south in Shades valley indicate that the ore-beds with their associated strata flatten out and locally rise towards the surface. The surface-rocks in the valley indicate irregularities in struc-



Irondale Seam: *A*, Morrow Gap; *B*, Wahanetta Mine; *C*, Olivia Mine; *D*, 0.75 mile SW. of Red Gap; *E*, Valley View Mine; *H*, Ruffner Mine No. 1.

Big Seam, workable part: *G*, Ruffner Mine No. 1; *I*, Helen-Bess Mine; *J*, T. C., I. & R. R. Co. Slope No. 12; *K*, T. C., I. & R. R. Co. Slope No. 8; *L*, Woodward Slope No. 1; *M*, Raimund Slope; *N*, Potter Slope No. 2; *O*, Sparks Gap.

Ida Seam: *F*, Ruffner Mine No. 1.

FIG. 7.—SECTIONS OF CLINTON IRON-ORE DEPOSITS IN ALABAMA.

ture, including faults, which would naturally be shared by the beds below.

No great deterioration in either quality or thickness of the hard ore in the direction of dip has yet been disclosed by the deeper slopes, an encouraging fact in so far as it can be used as a measure of the ore ahead of shorter slopes. At one of the larger mines, centrally located, systematic analyses are reported to have been made of the ore of the Big seam at intervals of a few feet from the outcrop to the bottom of the slope and throughout the extent of each entry to the right and left of the slope. The composition of the ore has been found to vary appreciably from place to place, and the degree of variation is likely to be as great within a few yards as it is between remote parts of the mine, but the average run of the hard ore in the mine is remarkably regular. It is stated that this series of analyses show that the content of metallic iron increases about 1 per cent. for each 1,000 ft. beyond the point where the ore becomes hard; that the lime (CaO) decreases about 1 per cent. in the same distance and that the silica-content increases a trifle. Slightly different facts are shown, however, by a series of analyses of ore from a mine also on the Big seam, NE. of Birmingham, and distant about 18 miles from the mine just mentioned. Here the lime is increasing slightly, while the insoluble material as well as the iron is decreasing slowly. This change is accounted for in all probability by the fact that the iron-ore here is still being mined from the zone of transition from soft to hard ore and that the completely hard ore has not yet been reached.

Studies by members of the Alabama Geological Survey extending over many years have shown that the Clinton formation tends to thin out and become sandier towards the SE. This change should be shared proportionately by the inclosed ore-beds, and the drill-records available indicate that such is the case. However, the number of complete drill-records available from the valley east of Red mountain is so few that no reliable conclusions can be based on them regarding the ore-beds in Shades valley. Ore, gradually diminishing in thickness to the SE., can, perhaps, be expected to underlie this valley probably to beyond Shades mountain. An estimate of the ore-reserves in Division A, based on these considerations, is given on pp. 126 to 130.

Division B.—Division B of the district comprises the strip of Red mountain from Sparks gap SW. to Big spring, a distance of 17 miles. Although of considerable extent, the actual mining of ore has been only a little stripping on the outcrop near Sparks gap and a short slope at Big spring, and but little prospecting has been done. That there is ore in the Clinton formation throughout this whole stretch of the mountain is beyond doubt, and it remains for more thorough prospecting to demonstrate whether or not it is of workable grade and quantity. Discouraging results have been obtained in places, but these have been widely separated, and cannot be regarded as conclusive evidence as to the character of the ore beyond the present limit of mining activity, especially since in most cases the prospecting was done only on the weathered outcrop of the beds. The most promising field for exploration in this division lies on the east slope of Red mountain and in the valley farther east, SW. of a line drawn at right angles to Red mountain through McCalla station and NE. of a line drawn similarly through Green pond.

Three beds of ferruginous material, probably corresponding to the Hickory Nut, Ida, and Big seams, have been observed, in places, with thicknesses respectively of about 15, 26, and 36 in., and the hard ore of the two lower beds carries from 25 to 40 per cent. of metallic iron.

Prospecting by means of a core-drill will be necessary in order to demonstrate whether or not there is an ore-field in this vicinity that can be opened up by shaft-mining. Such a field would be fairly accessible to existing transportation-lines, and not at all remote from furnaces.

So little is known concerning the quality and extent of the ore in the area of Division B that it would be useless to attempt an estimate of its ore-reserves. The best service that can be rendered by the forthcoming report is to show collectively what facts have been disclosed by prospecting, to represent by a geologic map the outcrop and relations of the Clinton formation, and to express the opinion stated above as to the location of the most promising field for exploration. Red mountain throughout Division B is a direct continuation of the ridge of richest ore-bearing strata in the State, and for part of the distance here the structure is apparently in its general features

similar to that of the most important part of Red mountain; that is, the rock-dips are, so far as could be ascertained, mainly moderate and regular to the SE., in the area between McCalla and Big spring. This feature is in itself an important consideration in relation to future prospecting, leaving the questions of quantity and quality of the ore yet to be determined.

Division C.—Division C includes that portion of Red mountain that extends from Morrow gap NE. about 6 miles, besides the area underlain by Clinton strata around the margins of the two synclinal basins along the upper part of Cahaba river, and the strip of Clinton that is exposed along the Alabama Great Southern railroad between Trussville and Springville. Altogether there is in this division a linear outcrop of Clinton strata approximating 28 miles, varying in width from 150 ft., where the measures are vertical and partly buried in a fault, to more than a mile, as in the shallow syncline 5 miles north of Trussville.

No underground mines have been opened in this area. Some soft ore has been obtained from surface-workings, especially in the vicinity of Red gap. Prospecting has been done on the outcrop of the beds, in many places, but especially within the first 6 miles NE. of Morrow gap, and it is probable that surface-mining of soft ore will soon be systematically developed in this locality. The Louisville & Nashville railroad extends in the valley between Red mountain and Little Sand mountain about 4.5 miles NE. of Red gap, and could readily build spurs 2 or 3 miles farther north if developments demanded further extension. Farther NE., however, the synclinal area between Big Cahaba creek and Little Canoe creek, and the one in the vicinity of Cahaba mountain, are not so well situated as regards present transportation-facilities, although there are no points which could not be reached by railway if sufficient ore is found to be there.

The most promising locality in Division C is that part of Red mountain northward from Morrow gap 4 or 5 miles. The workable ore-bed is here correlated with the Irondale. Its thickness ranges from 20 to 24 in. where it can be considered as of workable thickness at all, and there the workings will of necessity be confined to trenches on the outcrop and to open-cuts where the requisite stripping is not too heavy. As to

quality, the ore is, where soft, of fair grade, carrying from 40 to 54 per cent. of metallic iron; as to position, the bed in this locality is for the most part well situated with regard to the topography and to transportation-facilities, and its cover and dip are favorable to economical stripping and open-cut mining. Therefore it is probable that a large tonnage of soft ore is still available here.

Division D.—Division D includes West Red mountain from Compton, a small mining-town, 2 miles NNE. of Village springs, to near Tarrant gap. It comprises a single strip of Clinton formation about 15 miles long. Through the whole distance the Louisville & Nashville railroad traverses the valley SE. of the ridge, at a distance of from 0.5 to 1 mile from the latter. There is only one active mine in this division, the one at Compton. The rocks are nearly everywhere highly inclined on West Red mountain, and for 8 or 9 miles are bordered on the SE. by an overthrust fault.

On West Red mountain the rocks dip at high angles and appear not to carry valuable beds of iron-ore throughout the middle of the district. At the extreme ends of the district, however, the dips are more gentle, and workable beds have been discovered, for instance, at Compton and Dudley. Sections 5 and 6 of the Clinton, given on pp. 80 and 81, show the general character of the formation in the NE. half of West Red mountain. Workable ore occurs at Compton in but one bed, sections of which are given below.

Sections of Ore at Compton Mine.

First Right Entry, near Main Slope.		First Left Entry, 300 ft. from Main Slope.		Outcrop, near Top of Mountain.	
	In.		In.		In.
Shale.		Shale.		Shale, ferruginous.	
Ore, . . .	13	Ore, . . .	17	Ore, . . .	7
Shale, . . .	1 to 2	Shale, . . .	1 to 3	Shale, . . .	1.5
Ore, . . .	16	Ore, . . .	18	Ore, . . .	29
Shale.		Shale.			

The bed at Compton ranges generally from 30 to 36 in. in thickness, with a thin parting of shale, irregular in position, as shown in the sections. Locally, the entire bed is pinched down to a very few inches or entirely cut out by downward bulging of the overlying shale, which at such places has a con-

cretionary or concentric structure. Such structures, which result in the local disappearance of the ore, are termed "faults" by the miners, but there appears to be no dislocation of the beds and the ore is usually picked up again if the workings are driven on in the same plane.

Analyses of the hard ore at Compton mine show about 40 per cent. of metallic iron, 10 per cent. of insoluble material, and 15 per cent. of lime (oxide).

Field-examination of this part of the district indicates that in the strip from above Compton to Tarrant gap there are two beds of ore. The upper one is thin, from 1.5 to 3 ft. thick, but carries good hard ore in the upper end of the area, while the lower one is thicker, running from 5 to 11 ft. in thickness, but is nearly everywhere comparatively lean in iron and high in silica, so that it would be of value only locally, and then only as soft ore. Conditions for underground mining are favorable only in the upper end of the area; and, owing to the steep dip and heavy cover, open-cut work and trenching can be done only to a limited extent on the outcrop. On the other hand, the ore is everywhere within a short haul to a railroad, so that, if conditions should demand it, work might be done on a small scale in several places that will yield soft ore.

Division E.—Division E comprises the Clinton strata in West Red mountain and the fragmentary areas of the same within Birmingham valley NW. of Red mountain and SW. of Birmingham. These fragmentary areas include outliers within the chert-area of the valley, and strips like that of McAshan mountain and those in the extreme SW. end of the valley at Vance and SW. of Dudley. These last areas are almost completely buried by Cretaceous and Tertiary clays and sands. Roughly estimated, the total length of the outcrop of all the fragments placed end to end and including portions covered by Cretaceous and Tertiary sediments, but whose location is known, is 45 miles. The structure of the West Red mountain strip is similar to that in the same ridge above Birmingham, viz.: the strata stand nearly vertical or overturned, or rarely dip normally to the SW. In some places they are buried and in others offset greatly by faults. A structural feature not exhibited in any other of the divisions of the district is the duplication of Clinton areas by faulting and by folding with faulting. An

area duplicated by faulting is McAshan mountain; and the partly-buried ridges, extending from Vance and Dudley SW., have been formed by folding and faulting, forming unsymmetrical synclinal strips partly bounded by faults. Possibly of a similar type are the small remnants of Clinton strata that lie on the chert ridges NNE. of Bessemer.

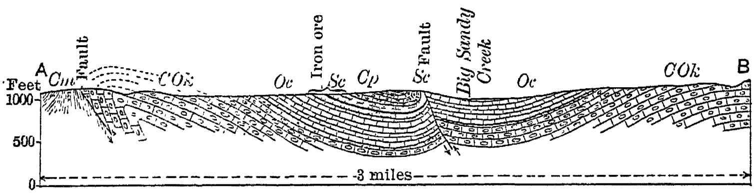
In July, 1906, there was one mine producing ore in this locality about 1.5 miles south of Dudley. A little mining has been done near the northern end of McAshan mountain, also in West Red mountain north of Woodstock, and at Vance, but work at these three places has long since been abandoned.

South of Bessemer the Alabama Great Southern railroad passes SW. through the valley from 1.5 to 2.5 miles from West Red mountain as far as Dudley, where it leaves the ore-bearing area and turns westward across the deeply-buried coal-field. At Chamblee the Louisville & Nashville railroad follows a gap through West Red mountain, crossing again to the east side of it in the gap eroded by Valley creek. Northeast of Bessemer the Louisville & Nashville railroad is again near West Red mountain. Therefore this division is well supplied with transportation-facilities, so that wherever the ore can be profitably mined, it can be reached easily by a spur from an ore-carrying road.

It has been stated that in Division E of the Birmingham red-ore district there are about 45 miles of outcropping ore-bearing strata. In reviewing the data obtained in the field it is not entirely certain whether any of this area contains ore-beds that can be depended upon to yield a profitable tonnage of good-quality ore. The beds have been worked at four places and prospected at scores of other places, but only at one place was mining active in 1906. According to the ore-sections that were measured, beds, mostly of lean to fair-quality ore, with thicknesses ranging from 2 to 6.5 ft., have been shown to be present throughout West Red mountain, McAshan mountain, and in one of the synclines at the SW. end of the area. According to analyses, the ores, mostly soft, carry percentages of iron ranging from 26 to 60 per cent.; of silica, from 60 to 10 per cent.; and of phosphorus, no higher than the average Red mountain ore. Therefore the lack of development in this area must be due to the unfavorable structure of the beds, which not only renders min-

ing difficult or impossible, but also makes very uncertain the extent of the ore beyond the outcrop. West Red mountain, with its highly tilted or overturned strata, nowhere offers good opportunities for slope-mining on the dip, and there are few places where the slopes or drifts can be driven on the strike of the seam. McAshan mountain has the same disadvantages, with the added one of limited extent. The small areas north of Bessemer have been mostly stripped of their available soft ore. It still remains for the locality SW. of Dudley to be thoroughly prospected before the scale of developments can be determined.

Encouraging results, so far as quality and thickness of ore are concerned, seem to have been obtained at the first slope south of the railroad, and the structural conditions indicate that there may be a uniform dip, perhaps reaching 20° to 25°



COk, Knox Formation. *Oc*, Chicamauga Limestone. *Sc*, Clinton Formation. *Cp*, Fort Payne Chert. *Cm*, Coal Measures.

FIG. 8.—STRUCTURE-SECTION ACROSS VALLEY OF BIG SANDY CREEK, 2 MILES SOUTH OF DUDLEY, ALA.

down the northwest limb of the syncline, for at least 0.5 mile. Beyond this distance there may have been a sudden backward and upward bending of the strata, possibly accompanied by such fracturing of the ore-beds as to render them of little value. This hypothetical condition is indicated in the structure-section, Fig. 8. The geologic relations in this vicinity must always remain very obscure, so far as surface-evidence is concerned, owing to the heavy mantle of coastal-plain sediments.

The development of a productive ore-field in this locality would be particularly fortunate for the district, since here the ore is within 3.5 miles of a coal-mine at Cedar cove in the Warrior coal-field. Both the coal and the ore are connected with the Alabama Great Southern railroad by short N–S. spurs from Dudley station, and there is a broad outcrop of undevel-

oped fairly pure Chickamauga limestone, suitable for fluxing-material, along the headwaters of Big Sandy creek, within a mile of the ore-mine, and within 3 miles of Dudley. Water-supplies can be provided by building storage-reservoirs on Big Sandy and South Fork Hurricane creeks. The establishment of an iron industry in the vicinity of Dudley would therefore be a natural result of this grouping of raw materials, providing the iron-ore proves to realize expectations. Geological conditions at least warrant thorough prospecting here.

A provisional estimate of the ore that should be contained in this portion of the district under certain conditions is given on p. 130.

Division F.—Division F comprises the areas of Clinton formation exposed around the southern margin of the Blount Mountain syncline and on the outlying knobs, such as Miles, Hayes, and Meridian mountains. Altogether this division occupies an area extending less than 5 miles from east to west and 3 miles from north to south. The general dips of the beds are gentle towards the NE., N., and NW., but the SW. rim of the syncline has been cross-faulted in at least three places, and the wedge-shaped block of strata in which Miles mountain stands has been literally squeezed out and dropped beyond the rim of the syncline, while an adjacent block containing parts of Butler, Hayes, and Meridian mountains has been uplifted. There is no transportation-line nearer to this area than the Louisville & Nashville railroad, which is at the nearest point about 1 mile from Miles mountain, and 5 or 6 miles by wagon-road from more-distant prospects.

No mining has been done in this locality, but prospecting has shown that there are two and possibly three distinct beds of ore in the formation in this locality. The top one is only 6 to 8 in. thick, the one next below is 2.5 to 3 ft. thick, while the lowest is, where recognized, thicker, but very siliceous. Analyses of soft-ore samples from these beds gave a range of from 33 to 51 per cent. of metallic iron, and from 45 to 15 per cent. of silica.

Topographic conditions affecting the accessibility of the ore are less favorable here than in other divisions of the district heretofore discussed. That the ore is of a doubtful grade is probable, since the hard ore in many of the outcrop-prospects

is very lean and calcareous. There is, of course, a possibility that some of the soft ore will be utilized eventually, since the ore-beds are to a large extent above water-level and can perhaps be mined on a small scale a short distance down the dip of the beds without requiring much initial expenditure. The beds are, for the most part, under cover too thick for stripping far from the outcrop, and this fact will discourage development. In any case there would be a long, and, in places, difficult wagon-haul to the railroad, and it does not seem that conditions warrant the building of a railroad-spur into the locality, although it would be entirely possible to construct one if the quantity and quality of the ores to be reached were found satisfactory.

Division G.—Division G includes the ore-bearing areas in the vicinity of Springville, St. Clair county, on the flanks of the upper part of Canoe Creek valley. There are two outcrops of the Clinton formation, one about 2.5 miles NW. of Springville, which is a NE. continuation of the SE. limb of the Blount Mountain syncline, the lower end of which is described in Division F. The other outcrop is just SE. of, and parallel to, the Alabama Great Southern railroad as far NE. as Springville, where it bends at right angles towards the SE. and extends on the NE. side of the valley of Right Hand Prong of Little Canoe creek nearly to Truss Mill. This is really an extension of Division C, and is about 9 miles long. The outcrop NW. of Springville extends NE. more than 15 miles, but only the portion as far NE. as Goodwin's mill is considered here, a strip about 8 miles long.

Judging from the results of a hasty field-examination, the Clinton area on the NW. side of Canoe Creek valley NE. of Springville can still be regarded as a possible source of future ore-supply, since the main bed appears to carry some ore of workable thickness, quality, and extent, besides being favorably situated with regard to topography and transportation-facilities. The bed formerly mined west of Springville carries in places about 3 ft. of soft ore which contains about 45 per cent. of metallic iron. Structural complications may be encountered, chiefly in the form of sharp folds, and at one place—namely, about 1 mile SW. of Steele station, the Clinton is buried in a fault for a short distance.

D. *Mining-Development.*[8]—There have been three stages in the development of the mines in the Birmingham district. The first stage consists of trenching the ore-beds along the outcrop on the crest or on the NW. slope of Red mountain, and of mining the ore from open-cuts on the SE. slope. The ore obtained in this way is mostly soft. This method of mining has been possible only where the overlying beds are not too thick to be stripped profitably. Most of the mines have passed beyond this stage, but at the Helen-Bess and the Green Spring workings this very profitable type of mining may still be seen.

The second stage of development combines the open-cut and incline with underground work. A very fortunate relation between the structure of the Big and Irondale seams and the topography of Red mountain exists in many places, particularly in the northern half of the district, wherever the dip of the Clinton strata is approximately the same as the SE. face of the mountain. This face is cut by narrow V-shaped ravines at intervals of from 0.5 to 0.75 mile, and on both sides of many of these hollows the two seams are exposed from the crest to the foot of the ridge. Inclined tramways are built on the flanks of the ravine, and when the outcropping ore has been surface-worked entries are driven in on the strike of the ore-beds from each side of the ravine and the ore is mined from the upper side of the entries. A cable-tramway may be operated by gravity or by power, depending on the side of the mountain on which the ore is to be delivered. At the Sloss-Sheffield Ruffner mine No. 1 the tracks of the railway which transports the ore to the furnaces are on the SE. side of the mountain, making it possible for cars loaded with ore going down the mountain to pull up the empties, but at the Valley View mine of the Birmingham Ore & Mining Co. the ore is hauled up over the mountain and loaded into railroad cars on the opposite side. At mines of this type, soft, semi-hard, and hard ores are obtained, depending on the thickness and character of the cover overlying the ore-bed.

The third stage of mining, the one to which the majority of the workings in the Birmingham district have now attained, involves systematic underground work entirely. The general

[8] See also Crane, W. R. Iron Mining in the Birmingham District, Ala., *Engineering and Mining Journal*, vol. lxxix., No. 6, pp. 274 to 277 (Feb. 9, 1905).

plan is very simple, comprising a main or central slope, driven on the dip, from which right and left headings are turned off at regular intervals of from 60 to 70 ft., as shown in Figs. 9, 10, and 11. The ore, which is mainly hard, is mined from the upper side of the entry, about 30 ft. being left between the entries until robbing is begun. Mules haul the trams to the mouths of the entries, whence the ore is moved up the slope by cable to a tipple, below which it is crushed and loaded directly into cars bound for the furnace. A manway is usually provided at one side of the slope for safety. Comparatively little water is encountered even in the deepest workings of this type, so that a 3- to 4-in. pump usually suffices to drain the mine.

A fourth stage, which some of the workings may reach in the near future, will likely be shaft-mining in the basin east of Red mountain. The working-face of the ore-bed can be reached more directly by a vertical shaft from 300 to 500 ft. in depth than by a slope five to six times that length.

In the summer of 1906, there were 33 mines actively producing red ore in the district, besides seven or eight workings which have been inactive since the soft ore was exhausted from them. Of the 33 mines in operation, 30 are on Red mountain, within a distance of about 25 miles between Pilot knob on the NE. and Sparks gap on the SW. In places in the middle of the district the underground workings are practically continuous for 3 or 4 miles, and the old surface-workings on the outcrop of the ore may be traced without break for 15 miles or more.

2. *Northeast Alabama.*

The discussion of the stratigraphy on pp. 76 to 82 applies to this area also, for Little Wills valley, along which workable Clinton ore outcrops, is almost a NE. continuation of the area described as Division G of the Birmingham District. Little Wills valley lies along the west side of Lookout mountain, and here the rocks dip generally from 10° to 30° towards the mountain, but locally there are dips as steep as 85°. On the east side the rocks dip steeply towards the mountain and in places are almost vertical. Along the NW. side of Little Wills valley lies a ridge known as Red mountain. Clinton strata form its crest and NW. flank, and dip SE. below the Fort

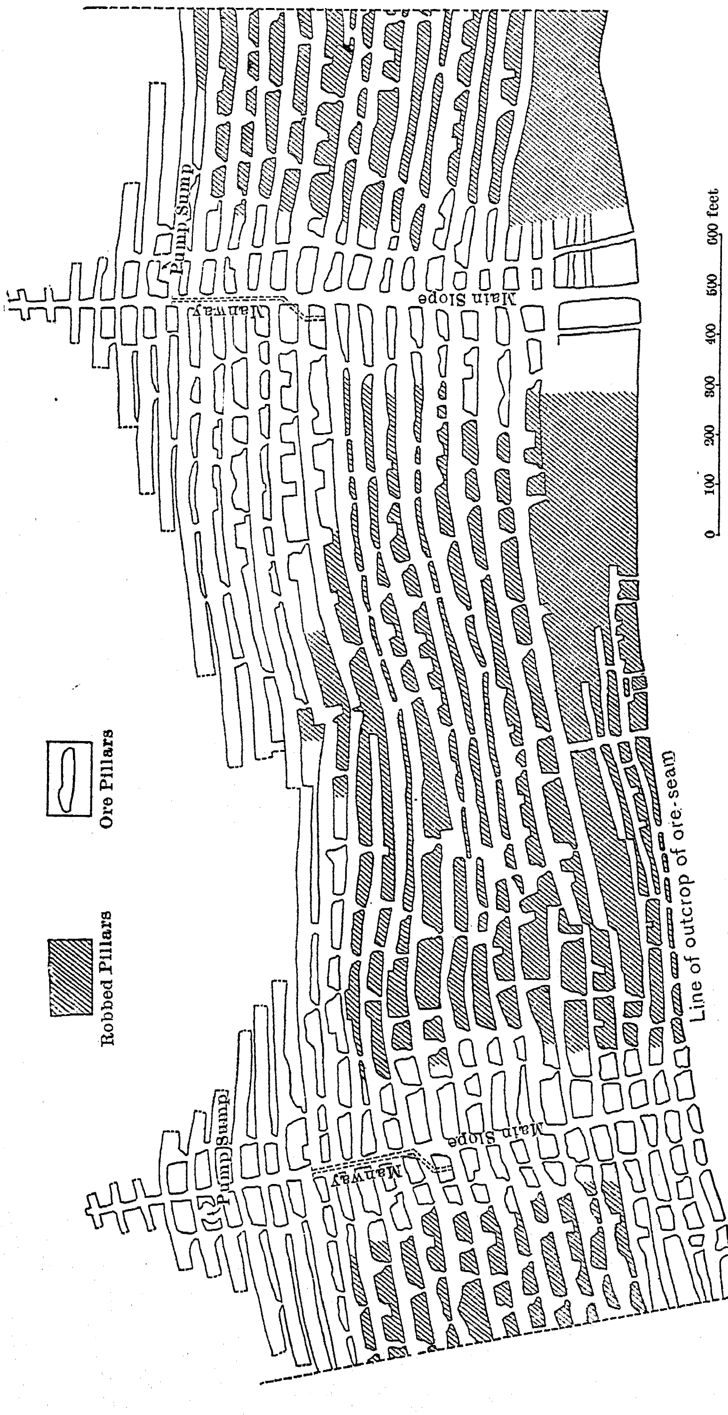


Fig. 9.—Typical Plan of Two Adjacent Slopes on Red Mountain, Driven Directly on Dip of Beds.

Payne chert, which, in turn, is overlain by Mississipian limestone in the valley. The Clinton formation, therefore, probably underlies the Lookout Mountain syncline, since it outcrops also along the east side of the mountain except in a few places where it has been dropped below the Knox dolomite by faults.

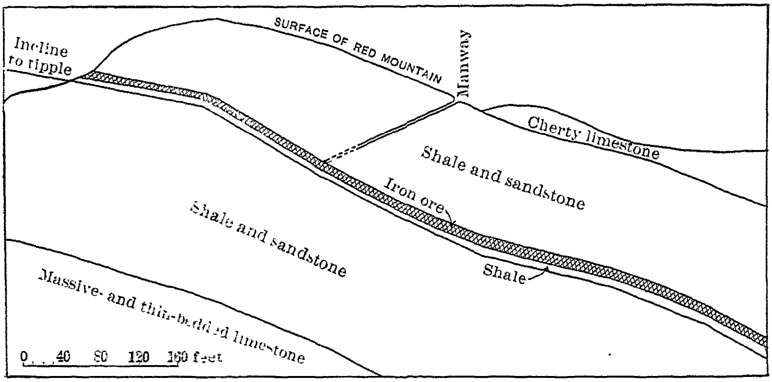
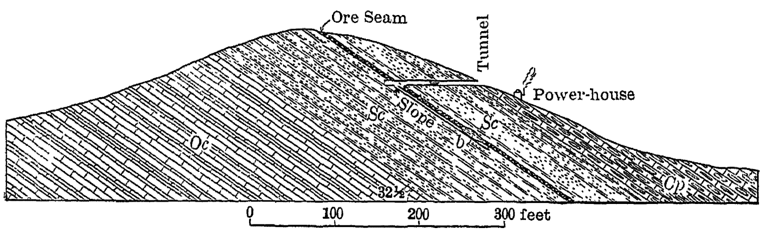


FIG. 10.—TYPICAL PROFILE OF SLOPE ON RED MOUNTAIN, STARTING ON OUTCROP.

The ore has been mined by open-cuts and short slopes at half a dozen or more places between the Georgia line and Attalla. One to four beds of ore are present, ranging in thickness from 1 to 4.5 ft., and rarely thickening locally to 7 ft. At Battelle there are four seams of ore, two of which are mined



Oc, Chicamauga Limestone. *Sc*, Clinton Formation. *Cp*, Fort Payne Chert.

FIG. 11.—PROFILE OF SLOPE REACHED BY TUNNEL.

from slopes. The uppermost is 4.5 ft. thick and dips from 23° to 45° SE. The hard ore carries from 23 to 31 per cent. of metallic iron, from 7 to 16 per cent. of insolubles, from 22 to 26 per cent. of lime, and 0.38 per cent. of phosphorus. The bottom bed, about 3 ft. 4 in. thick, dipping about 85°, is mined from almost vertical openings. Its ore carries about 32 per

cent. of iron, 10 per cent. of insoluble, and 21 per cent. of lime.

At Fort Payne one bed is mined, the section of which shows two benches of ore, each about 2.5 ft. thick, parted by from 1.5 to 2.5 ft. of shale. The ore dips about 14° SE. The hard ore carries about 26 per cent. of iron, 4 per cent. of silica, 30 per cent. of lime, and 0.36 per cent. of phosphorus. The soft ore shows 56 per cent. of iron, from 8 to 10 per cent. of silica, 1 per cent. of lime, and about 0.45 per cent. of phosphorus.

At Portersville one bed of ore is mined that shows a parting of shale in the middle similar to that at Fort Payne, but the total thickness of ore is only about 4 ft. Here the ore dips about 15° SE. It carries an average of 32 per cent. of iron, 6.3 per cent. of insolubles, and 0.25 per cent. of lime. A second ore-bed 1 ft. thick is present about 50 ft. below the main ore-bed at Portersville.

Until recently there was mined at Crudup a bed of ore locally thickening to 5 or 6 ft. with one or more shale partings. The soft ore carries about 43 per cent. of iron, 10.5 per cent. of silica, and about 0.44 per cent. of phosphorus. Semi-hard ore from here carries 39 per cent. of iron, 10.5 per cent. of silica, 3.3 per cent. of alumina, 14 per cent. of lime, and 0.42 per cent. of phosphorus.

Near Attalla, a bed of ore from 3 to 4 ft. thick has been mined down a slope for about 1,000 ft. without marked change in character. The beds dip from 30° to 35°. The soft ore carries from 51 to 54 per cent. of iron, from 8 to 12 per cent. of insoluble material, and 0.68 per cent. of phosphorus. The hard ore carries from 38 to 44 per cent. of iron, 9 per cent. of silica, and from 18 to 27 per cent. of lime. Southwest of Attalla this strip of Clinton formation is offset to the west by faults.

On the east side of the Lookout Mountain syncline, near Gadsden, the ore is worked at two places from slopes on a single bed in the SE. face of Shinbone ridge. The ore ranges from 2.5 to 4 ft. thick, and dips steeply, being probably overturned in places. The hard ore is reported to carry 34 per cent. of iron, 5.5 per cent. of silica, and 19 per cent. of lime.

On Dirtseller ridge the Clinton ore is exposed in places from Gaylesville NE. into Georgia. Near Gaylesville open-

cuts and slopes have exposed the ore for 2 miles along the strike, on the NW. limb of the Dirtseller syncline. The ore is generally thin, from 20 to 24 in. being the usual range, although it thickens locally to 3 ft. or more and in places thins down to a very few inches. The soft ore carries about 50 per cent. of iron, 20 per cent. of insolubles, 2.5 per cent. of lime, and 0.44 per cent. of phosphorus.

Southwest from Gaylesville about 15 miles stands Round mountain, a small outlier of the west limb of the Dirtseller syncline. Clinton ore has been mined here at irregular intervals since before the Civil War, from a bed about 3.5 ft. thick having two benches of ore, each from 14 to 17 in. thick, parted by a shale-seam from 12 to 15 in. thick. The ore dips irregularly from 10° to 45° to the SE. The upper of the two benches contains the softer and the more fossiliferous ore. The lower bench is somewhat oölitic, and often has, when freshly broken, a steely, metallic luster. The soft ore is reported to carry 52 per cent. or more of metallic iron.

In summarizing the available data it is found that the total length of outcrop of the Clinton formation in the portion of NE. Alabama here described is about 100 miles. This does not include any of the outcrops of Clinton exposed on the west limb of the anticline lying west of the Lookout Mountain syncline, and none of the Clinton along the anticline in which flows the Tennessee river from the Tennessee line SW. to Guntersville, because in those areas no ore is known to be workable under present conditions.

Of this 100 miles of outcrop, 50 miles are along the west border of the Lookout Mountain syncline. Not more than 25 miles of this outcrop may safely be counted as having workable ore, but where the ore is workable it may be expected to have a fairly-wide extent below that mountain. A sufficiently close study has not, however, been made of the structure of this area to make any safe predictions as to the attitude of the beds very far beyond the ends of the present slope-mines, the longest of which is about 1,000 ft. The ore in the main seam averages, according to measurements where mined, a little over 4 ft. thick, exclusive of shale partings. None of the ore-beds have been correlated with those in the Birmingham district. The hard ore here is rather lean, and may not be found to

average more than 25 per cent. of iron, if the whole area be considered. A slightly better grade of ore is found on the east side of the Lookout area, but the bed is thinner, 3 ft. probably representing a liberal average for its thickness. Structural conditions are plainly less favorable here, since the ore is more or less steeply inclined and faulted. The extent of the outlying areas of ore, such as Dirtseller ridge, is not large, but the accessibility of the ore makes it probable that such areas will still be important contributors to the Gadsden, Rome, and Chattanooga blast-furnaces.

IV. Possible Origin of the Ore.

E. C. Eckel has summarized the theories regarding the origin of these ores as quoted below. The discussion that follows is adapted, with modifications, from Mr. Eckel's contribution to a paper now in preparation.[9]

"*Theories of Origin.*—For many years the origin of the oölitic and fossil ores which occur in rocks of Clinton age has been a much discussed subject. Disregarding minor points of difference, it may be said that the various theories which have been advanced to account for the origin of these ores can be reduced to three. These three opposing theories are, briefly stated, as follows:

"(1) *Original Deposition.*—The ores were formed at the same time as the rocks which inclose them, having been deposited in a sea or basin, along with the limestones, sandstones, and shales which now accompany them.

"(2) *Residual Enrichment.*—The ore-beds are merely the weathered outcrops of slightly ferriferous limestone, the lime having been leached out above water-level, leaving the insoluble portion of the limestone in a concentrated form.

"(3) *Replacement.*—The ores are of much later origin than their inclosing rocks, having been formed by the replacement of original beds of limestone by iron brought in by percolating waters.

"*Practical Importance of the Question.*—In addition to the questions of purely geologic interest which are connected with the differences of opinion as to the origin of the Clinton ores, the matter has a very distinct practical bearing on the working of the ores. This phase of the subject may be stated as follows:

"If the ore-deposits are due to the replacement or surface-decay of a limestone, they can be expected to decrease rapidly in value in depth, becoming lower in iron and higher in lime, until at no great depth the bed will consist entirely of unaltered limestone.

"If, on the contrary, the ore-deposits are original, no such regular decrease in richness is to be expected as the mines are driven deeper. Patches of low-grade ore may be struck, but these will be due to original differences in the richness of the ore, and a slope might pass downward through such a patch of lean ore into another area of high-grade ore."

---

[9] *Bulletin of the U. S. Geological Survey* No. —, The Iron-Ores, Fuels, and Fluxes of the Birmingham District, Ala. (in preparation).

A sufficient array of facts has been obtained in the course of work on Alabama Clinton iron-ore to render support to the theory of original deposition by Smyth,[10] and to offer serious objections to the residual-enrichment and replacement theories.

The residual-enrichment theory, requiring that the iron oxide shall decrease and the lime carbonate increase as the ore-bed is followed down from its outcrop, accounts perfectly for the existence of soft ores along the outcrop, and down the dip to the limit of the zone of weathering. As the ore becomes more calcareous the thickness may become perceptibly greater, although in the case of the soft ore the latter is under less pressure, is less dense, and therefore may occupy nearly if not quite as great a thickness as the calcareous ore from which it has been derived. The theory, however, requires that where the level of ground-water is reached the place of the ore-bed shall be occupied by a much thicker bed of limestone. Experience in mining the ore in every slope in Alabama that has passed beyond ground-water level shows that the ore does not give place to a thicker bed of slightly ferruginous limestone, but rather to an almost uniformly calcareous ore, whose richness apparently depends on the quantity of iron originally included in the sediments. The theory would require, moreover, that a very great shrinkage must have taken place in the secondary concentration from a ferruginous limestone to an ore. Thus, to produce a bed of ore such as the Big seam of the Birmingham district, making due allowance for the low grade of much of its total thickness of 30 ft., would require originally at least 8 ft. of limestone to 1 ft. of ore, or 200 ft. of limestone to 30 ft. of ore. The inevitable structural effects of such a shrinkage are, however, lacking, even in the deepest mines.

Certain of the requirements of the replacement theory are: (1) that as the ore-bed is exploited deeper and deeper the content of iron oxide shall steadily give place to calcium carbonate, until only a ferruginous limestone is reached; (2) the iron oxide shall be found to replace limestone strata, particularly where opportunities were favorable for the passage of iron-bearing waters, such as along fault-zones and in badly disturbed, brecciated strata; (3) that a bed of ore overlying

[10] Smyth, C. H., Jr. On the Clinton Iron Ore, *American Journal of Science*, Third Series, vol. xliii., No. 258, pp. 487 to 496 (June, 1892.)

an impervious shale shall be richer in iron at its base and richer in lime carbonate at the top; (4) that red ores of this character should have a wider vertical range, and replace Silurian limestones outside of the Clinton formation, and perhaps also those of the Ordovician, Carboniferous, and Devonian systems. None of these conditions, however, are found to be fulfilled.

If either process had contributed to any great extent to the formation of the present Clinton ore-beds, the iron probably first existed in the carbonate form before becoming the oxide, and it is difficult to explain the change of iron carbonate directly to the anhydrous oxide, and also how the oölitic concretions of ore inclosing grains of sand were formed under these conditions.

The principal facts supporting the theory of sedimentary origin may be briefly summarized as follows:

(1) In mining from slopes running down on the dip of the ore-bed, when once the limit of surface-weathering is passed—and this may occur at any point from 1 to 400 ft. below the outcrop—no further regular change in the ore is found with increasing depth, though a number of mine-workings are now close to 2,000 ft. from the outcrop. Areas of low-grade ore and of barren shale have been encountered, but such areas are plainly the result of original deposition. Mine-stopes have been driven onwards through such patches of lean ore or rock into ore of good grade.

(2) A number of borings in Alabama have struck the ore at points from 0.5 to 1 mile back from the outcrop, and at depths of from 400 to 800 ft. below the surface. The ore encountered in these borings was hard ore of the usual quality, and not merely a ferruginous limestone. Several borings in New York are reported to have struck Clinton ore at distances of from 10 to 15 miles back from the outcrop, and to have shown good hard ore at depths of from 644 to 955 ft. below the surface.

(3) The physical character of the oölitic ore cannot readily be explained on any replacement theory, while the formation at the present day of original oölitic materials is known to be taking place.

(4) The occurrence of fragments of the ore in an overlying

bed of Clinton limestone, as mentioned by Smyth,[11] points to the fact that the ore had been formed prior to the deposition of this limestone.

The strike-sections of the ore-beds exposed in Alabama indicate that the beds are lens-shaped, and where the lenses thin they tend to become shaly and sandy. Many ore-sections show shale partings similar to sections of coal-beds, and the transition from ore to the inclosing sandstone or shale is usually as sharp as that between coal and its inclosing rocks. These relations suggest at least a similarity in the mechanical methods of concentration of coal and of ore-sediments by bodies of shallow water. The formation of the oölites and the replacement of calcareous fossil fragments by iron oxide may have taken place on the sea-bottom prior to consolidation of the rock-beds.

V. Relation of Origin to Quality and Extent of Ore.

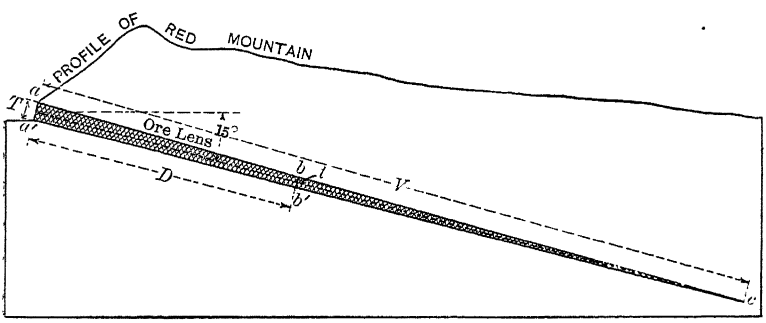
Referring to the theories regarding the origin of the ore outlined on p. 119, it has been observed that the ore, where soft at the surface, changes more or less gradually into hard ore with depth. This fact indicates clearly that the soft ore has been derived by a secondary process from the hard ore, but it does not furnish any suggestions concerning the genesis of the hard ore itself. The mode of occurrence and the constitution of the hard ore do not indicate that it has resulted from the alteration of a rock originally very different in composition, or that it is directly residual from disintegration of rocks containing minor quantities of iron-minerals. The hard ore must therefore be regarded as having been formed in essentially its present condition, contemporaneously with the inclosing sandstones and shales of the Clinton formation. Acceptance of this view leads to the conclusion that no regular decrease in iron-content is to be expected as the ore-beds are explored to greater depths than those already attained.

As to the extent of the beds, observations in mines and along the outcrops, as well as general studies of the stratigraphy, show that the ore-beds, in common with the other strata in the Clinton formation, are built up of overlapping thin lenses or

---

[11] *American Journal of Science*, Third Series, vol. xliii., No. 258, p. 493 (June, 1892).

layers of sedimentary materials. As a whole, the Clinton formation exhibits this lenticular structure, and therefore the ore-beds also probably originally were lenticular in shape, and the ore-lenses were comparable in length and width with sandstone and shale lenses in the Clinton formation. The lenses were, of course, very much flattened, and very thin in proportion to their other dimensions, and they probably thinned to a feather-edge in some directions, while in others they split into thin seams and dove-tailed with lenses of sandstone and shale near their extremities. The lenses of sandstone, shale, and ore composing the Clinton formation were probably deposited in a nearly horizontal attitude, or at least with a low initial dip. Folds, faults,



V = Average distance from outcrop at which ore-bed becomes very thin.
T = Average thickness of ore-bed at outcrop.
t = Minimum thickness practicable to work ore.
D = Distance from outcrop at which thickness of ore-bed becomes t, or maximum distance practicable to drive slopes.

FIG. 12.—DIAGRAM ILLUSTRATING THINNING OF AN ORE-BED.

and erosion have so tilted, broken apart, and worn away portions of the rocks that it is difficult to recognize in the present outcropping beds portions of what probably were originally well-defined, lens-like bodies of ore.

Considering in this light the ore-beds exposed along Red mountain in the heart of the Birmingham district, it may be possible that the Big seam, the Irondale, and other minor beds represent portions of thin, narrow, elongated lenses of ferruginous rock. These beds are known to change in character only gradually along the NE–SW. line of exposures, which is in general parallel to the shore-line of the Clinton sea or embayment. . At right angles to this line, however, the changes would naturally be more abrupt, and since the beds have not

been found to become thicker to the SE., as they dip under the Carboniferous rocks, it is fair to assume that at a distance from the outcrop the ore-beds will begin to thin out, or to split and become shaly or sandy, and finally to become so thin or to so deteriorate as to become negligible from the stand-point of workability. The beds would thus form long, thin, wedge-shaped bodies, the thicker portion of the wedge lying along the outcrop on Red mountain, with the thin edge, somewhat less regular in outline, lying below the surface to the SE., as shown by the dot-and-dash line in Fig. 13. The approximate position of this line, which represents what may be termed the "vanishing-point" of the ore, is necessarily very uncertain. An ore-seam a few inches thick is brought to the surface by a fault east of Cahaba river. Other structural conditions, and drill-records, however, indicate that this "vanishing-point" cannot safely be placed farther east than Shades mountain, the border of the heavy cover of Coal-Measure rocks, and it is an open question whether it should be carried that far. Since there are several beds of ore in the formation, the maximum distance from the outcrop to the feather-edge of the lens would probably apply to the thickest and most persistent bed of the series, viz., the Big seam. The other, smaller beds, such as the Irondale and the Ida, would not be expected to continue so far, to judge from their extent and the relations exhibited along the strike, although the possibility of their overlapping the Big seam and thus continuing beyond it in the direction of the dip must not be overlooked. If it be desired to estimate the tonnage of possibly recoverable ore in one of these lens-fragments it is necessary to calculate the cubic contents of the ore-body as though it were projected in a plane instead of being bent and crumpled, as is probably its actual condition in many places. The effect of gentle foldings should not seriously alter the calculations, although sharp folding and faulting would necessitate throwing out of consideration practically all the ore involved. If there is a *locus* containing points beyond which the ore continues only a few inches in thickness, it will, of course, not be practicable to mine the ore to these points, and the limit to which it will pay to drive slopes will be determined by the minimum thickness of ore that can be mined with profit at a given depth, or distance down the slope (assuming always that

structural conditions are favorable, and that there is a fairly-regular decrease in thickness of the ore-beds from their outcrop towards the thin edge of the lens). Keeping all these possibilities in mind, and using certain additional data suggested below, the engineer or geologist should be able to make a fairly close estimate of the tonnage of the Clinton ore-reserves in a district or in any portion of it. The smaller the area involved in the estimate, and the more data available, the smaller the error will be.

VI. Estimates of Ore-Reserves.

1. *Method of Making Estimates.*

First, an area should be divided into units, somewhat after the manner in which the Birmingham district was divided, as outlined on p. 95. Then each division should be subdivided again and again until areal units are obtained in which the dimensions can be accurately measured. Every possible item of information concerning thickness of beds in outcrops, mines, drill-holes, etc., should be considered, especially in relation to actual locations, in order that errors introduced by too-general averages may be avoided. The percentage of recoverable ore should enter into the calculation, as well as the specific gravity of the hard ore based on a conservative average percentage of metallic iron.

If it be assumed, then, that the ore in a certain bed within a given area forms a fairly regular prism, the base and altitude of which may be measured, and that the minable ore of this seam constitutes a truncated portion of this prism, as shown in Fig. 12, the cubic contents of this truncated prism of minable ore may be calculated conveniently by substituting in a formula the values of the average thickness, length, and width of the truncated prism of ore. From this result (in cubic feet) may be deduced in the same operation the tonnage of ore of a definite grade by use of the factors, percentage of recoverable ore and specific gravity of the same, based on the average percentage of metallic iron in the hard ore. Multiplying by 62.4, the weight in pounds of a cubic foot of water, will give the pounds of ore, which can then be reduced to long tons by dividing by 2,240.

Therefore, to establish a general formula for calculating the ore-content for a certain ore-bed in a given area, let—

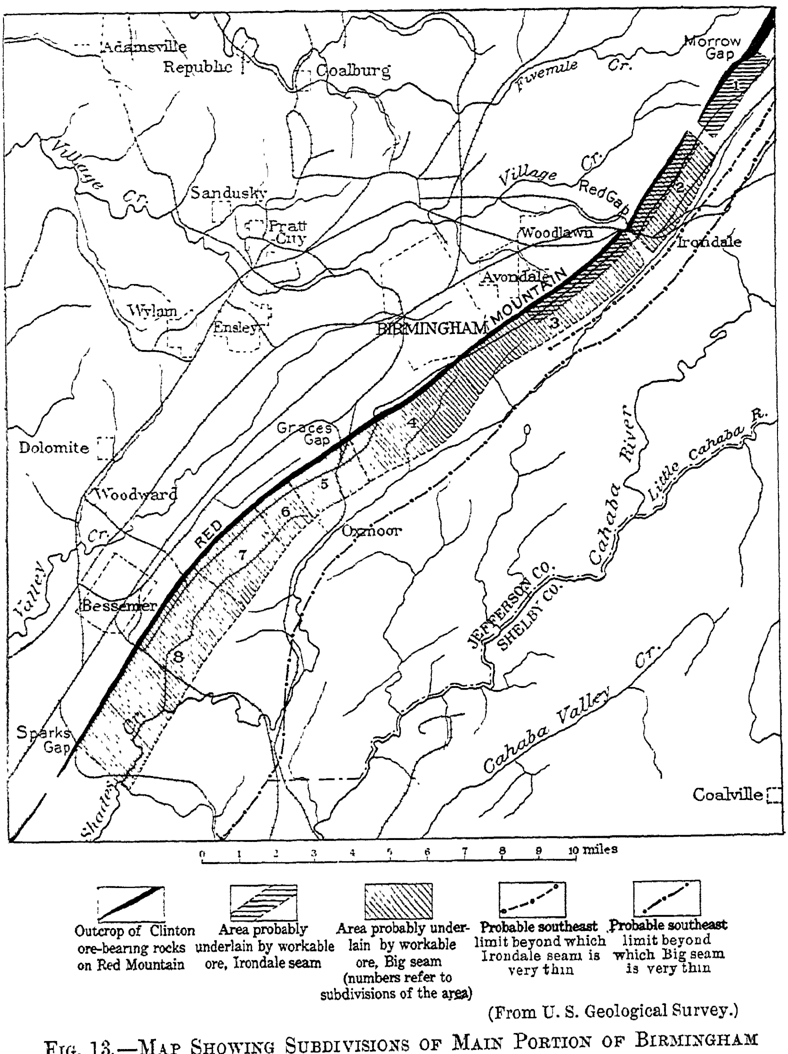
L = Length of outcrop.
V = Average distance of "vanishing-point" from outcrop.
T = Average thickness of ore-bed at outcrop.
t = Minimum thickness to which ore may be worked.
D = Distance from outcrop at which thickness of ore-bed becomes t, or maximum distance practicable to drive slopes on the dip.
R = Percentage of recoverable ore.
Fe = Average percentage of metallic iron in hard ore.
G = Specific gravity of ore based on value of Fe.

Then, to obtain the value of D in terms of the known quantities T, t, and V, in the similar triangles $a\,a'\,c$ and $b\,b'\,c$ (see diagram, Fig. 12), the base $a\,a'$ (T) : the base $b\,b'$ (t) : : the altitude V: the altitude $V - D$, or $tV = T(V - D)$. Then $tV = TV - TD$, and $TD = TV - tV$, whence $D = \frac{TV - tV}{T}$ and the total tonnage is $\frac{\frac{1}{2}(T + t) \times L \times D \times R \times G \times 62.4}{2{,}240}$.

2. *Data and Results.*

Estimates of the ore-reserves in Division A, and in part of Division E, of the Birmingham district, and in Little Wills valley in NE. Alabama, have been made in preparing this paper, but it is here emphasized that while many more details have been considered than there is space to enumerate here, the estimates must be regarded as only very tentative ones. For Division A there was computed the tonnage of ore that should be contained under the assumed conditions, first in the Irondale seam, from Morrow gap to Clifton gap; and second, in the Big seam, from Morrow gap to Sparks gap. From the sum of these estimated quantities was subtracted the total tonnage of red ore that has been produced in Alabama from 1880 to 1907 inclusive. In making this estimate, Division A is subdivided into eight parts, in two of which the Irondale seam is considered of sufficient importance to be regarded as a source of future ore-supplies. These eight units of area, shown in Fig. 13, whose ore-bearing strata outcrop along Red mountain, are as follows: (1) From Morrow gap to and including the Olivia mine (Irondale seam); (2) Bald Eagle to Clifton gap (Irondale seam); (3) Bald Eagle to

Lone Pine gap (Big seam, upper bench); (4) Lone Pine gap to Graces gap (Big seam, upper bench); (5) Graces gap to a point beyond Ishkooda (Big seam, upper bench);



(From U. S. Geological Survey.)

FIG. 13.—MAP SHOWING SUBDIVISIONS OF MAIN PORTION OF BIRMINGHAM DISTRICT, ON WHICH ESTIMATES OF ORE-TONNAGE ARE BASED.

(6) Ishkooda to slope No. 10 of the Tennessee Coal, Iron & Railroad Co. (Big seam, upper bench); (7) slope No. 10 of the Tennessee Coal, Iron & Railroad Co. to middle of Woodward Iron Co. property (Big seam, upper bench); (8) middle

of Woodward Iron Co. property to Sparks gap (Big seam, upper bench). The estimate is considered to be conservative for the following reasons: (1) No account has been taken of any possible available ore except that in Red mountain; (2) no ore-seams besides the upper bench of the Big seam and the Irondale seam have been considered; (3) only such portions of the outcrop of these seams have been considered as are known to be workable, and wherever the seams are faulted out or badly broken up, such portions are not included in the area upon which estimates are based; (4) the percentage of recoverable ore has apparently been placed low enough to be on the safe side; (5) conservative figures have been used as representing the average workable thicknesses at the outcrop, and the minimum workable thickness, since, under favorable conditions, the former may be considerably greater and the latter may be less, thus increasing the value of *D;* (6) the percentage of metallic iron used as a factor in determining the specific gravity of the hard ore has been taken with a view to the possible reduction rather than increase of iron-content with distance from outcrop; (7) in deducting the tonnage of red ore already produced the total red ore for the State has been taken, which is greater than that produced by the Birmingham district, and consequently in excess of that produced by this area, the main portion of the Birmingham district. In regard to this last factor, it should be stated that the excess is not great, however, for the Birmingham district has produced almost 90 per cent. of the red ore of the State; and Red mountain, between Morrow gap and Sparks gap, has produced between 97 and 98 per cent. of the red ore of the district. In 1907 the Birmingham district produced 89.3 per cent. of the total production of red ore in Alabama.

In obtaining the specific gravity of the hard ore in relation to its content of metallic iron, use has been made of the laboratory-determinations of R. T. Pittman, Chief Chemist of the Sloss-Sheffield Steel & Iron Co., at Birmingham. The experiments consisted of grinding lumps of typical Birmingham hard red ore down to 1-in. cubes, determining the specific gravity of each by displacement of water, and afterwards analyzing the ore thus treated. The results of certain of these tests and analyses are given in Table III.

TABLE III.—*Specific Gravity Tests and Analyses of Calcareous Hematite.*[a]

	Weight in Air 1 Cu. Ft. of Ore.	Specific Gravity.	Analyses.		
			Fe.	Insoluble.	CaO.
	Pounds.		Per Cent.	Per Cent.	Per Cent.
(1).......	213.47	3.42	36.25	13.80	17.98
(2).......	215.97	3.46	37.05	12.40	18.14
(3).......	219.23	3.50	57.60	11.42	17.43
(4).......	220.71	3.53	38.05	10.60	17.52
Average	217.35	3.48	37.24	12.05	17.78

[a] Experiments by R. T. Pittman, Birmingham, Ala.

While these experiments afford no direct data as to the porosity of the ore, nor as to its content of moisture, they do afford, by comparison with the calculated specific gravities, a constant factor of difference which is due to the effects of

TABLE IV.—*Estimated Red-Ore Reserves in Main Portion of Birmingham District.*

Sub-division.[a]	*L.*	*V.*	*D.*	*T.*	*t.*	*R.*	*Fe.*	*G.*	Total.
	Ft.	Ft.	Ft.	Ft.	Ft.	Per Ct.	Per Ct.		Long Tons.
1........	11,500	8,000	2,720	4.54	3.0	80	32.0	3.31	8,698,800
2........	32,000	6,800	2,000	4.24	3.0	80	35.0	3.40	17,554,600
3........	37,000	10,600	6,235	8.5	3.5	80	35.0	3.40	104,878,600
4........	16,500	12,000	7,300	9.0	3.5	80	33.0	3.33	55,865,000
5........	10,000	12,000	6,750	8.0	3.5	80	34.0	3.37	29,148,600
6........	6,000	13,000	7,167	7.8	3.5	80	36.0	3.42	18,517,800
7........	12,000	16,000	9,656	8.23	3.5	80	36.6	3.44	52,055,000
8........	29,500	24,000	9,500	8.32	5.0	60	36.8	3.45	107,624,400

Grand total by subdivisions before deducting production previous to 1908, ...	394,342,800
Deducting production 1880 to 1907 inclusive, . . .	43,193,300
Total red-ore reserves in main portion Birmingham district,	351,149,500

[a] (1) Irondale seam, Morrow gap to beyond Olivia mine.
(2) Irondale seam, Bald Eagle to Clifton gap.
(3) Big seam, upper bench, Bald Eagle to Lone Pine gap.
(4) Big seam, upper bench, Lone Pine gap to Graces gap.
(5) Big seam, upper bench, Graces gap to beyond Ishkooda.
(6) Big seam, upper bench, Ishkooda to Tennessee Co. mine No. 10.
(7) Big seam, upper bench, Tennessee Co. mine No. 10 to middle of Woodward property.
(8) Big seam, upper bench, middle of Woodward property to Sparks gap.

porosity and moisture. This ore, it must be remembered, is a very hard, compact material as mined, and in its normal condition underground carries very little moisture. The moisture present ranges generally between 0.5 and 1 per cent., and it rarely rises above 2 per cent. By using the average factor of difference, obtained as suggested above, a consistent specific gravity for any sample of hard ore of this district can be calculated if the content of metallic iron be given.

On applying the formula outlined on p. 126 to the area contained in Division A of the Birmingham district, the data and results shown in Table IV. are obtained.

If the hard ore in the vicinity of the new slope in Division E south of Dudley proves to be of workable grade, carrying 32 per cent. or more of iron, for 10,000 ft. along the strike, to maintain this quality, and a thickness of 6.5 ft. at the outcrop, and to decrease to not less than 3 ft. thick at 2,100 ft. on the dip, there should be an ore-reserve here of nearly 7,000,000 long tons, or enough red ore to supply the furnaces of the Birmingham district for two years at their present rate of consumption. It should be understood, however, that the presence of such a tonnage has by no means been demonstrated yet. For this portion of the district south of the Alabama Great Southern railroad at Dudley, the data and results are:

L.	*V.*	*D.*	*T.*	*t.*	*R.*	*Fe.*	*G.*	Total Ore.
Ft.	Ft.	Ft.	Ft.	Ft.	Per Ct.	Per Ct.		Long Tons.
10,000	3,960	2,132	6.5	3	75	32.0	3.31	6,929,600

For the ore-bearing area in NE. Alabama only a very vague estimate can be made, since I am not thoroughly familiar with the region, and the data at hand are not sufficiently complete or definite. Simply as a matter of interest in this connection, it will be shown what an enormous tonnage of low-grade ore (30 per cent. of metallic iron) would be contained in a single bed on the west border of the Lookout Mountain syncline, assuming that, in the aggregate, the length of outcrop that might be worked equals 25 miles, the average thickness at the outcrop is 4 ft., the minimum workable thickness is 2.5 ft., and the distance to the feather-edge of the ore-bed is 10,000 ft. From what is known concerning this area, it is thought that

these figures are entirely within the range of possibility, although no opinion is expressed here as to how profitable it might be to mine on a large scale a 30-per cent. ore from a 2.5- to 4-ft. seam as compared with the thicker, richer ores of the Birmingham district. The figures are as follows:

L.	*V.*	*D.*	*T.*	*t.*	*R.*	*Fe.*	*G.*	Total Ore.
Ft.	Ft.	Ft.	Ft.	Ft.	Per Ct.	Per Ct.		Long Tons.
132,000	10,000	3,750	4	2.5	75	30	3.24	125,651,100

Summarizing these results, there are estimated red-ore reserves:

	Long Tons.
For main portion of Birmingham district, ...	351,149,500
For district south of Dudley, ...	6,929,600
For Lookout Mountain district, ...	125,651,100
Total, ...	483,730,200

This brings the tòtal fairly well towards 500,000,000 long tons of ore, and it is probable that that amount would be exceeded by any estimate that considered carefully the portions of the field not included within the present estimate. The preliminary estimate made by Mr. Eckel[12] of 1,000,000,000 long tons of red ore in Alabama, including as it did much ore probably carrying from 25 to 30 per cent. of metallic iron, and occurring in beds at present regarded as too thin to be profitably worked, but of possible future value, appears to be consistently supported by the present estimate of ore-reserves probably workable under present conditions.

It should be repeated, in conclusion, that the present estimate is based on the belief that the hard-ore beds are the result of a single concentration of iron oxide sediments that took place when the beds were deposited; that they occur as fragments of what were originally rather uniform lens-shaped bodies; that, as a consequence of their supposed method of origin, the content of metallic iron does not greatly diminish from the point where the hard ore is first encountered in the mine-slopes to the point where the bed has thinned to a minimum workable thickness; and finally, that the structure re-

[12] Eckel, E. C. A Review of Conditions in the American Iron Industry, *Engineering Magazine*, vol. xxx., No. 4, pp. 518 to 527 (Jan., 1906.)

mains fairly constant, as indicated in the foregoing discussions. This last element, it should be remembered, is one of the most uncertain with which the miner has to deal, and can be rendered more certain only by thorough and systematic prospecting with the core-drill in places where there are no definite mine-data or reliable geologic indications available. Unexpected structural complications, and "horses" of barren rock may, of course, greatly reduce the quantity of recoverable ore counted on in this estimate.

VII. Production and Consumption of Iron-Ore.

Since 1894, Alabama has held third place among the iron-producing States. In 1907 the total production of iron-ore amounted to 4,039,453 long tons, composed of 3,144,011 tons of red hematite and 895,442 tons of brown hematite.

The Birmingham district, in 1907, produced 2,709,934 tons of red hematite, or 89.3 per cent. of the total tonnage of red ore in Alabama. The series of mines known as the Red Mountain group are classed among the prominent iron-ore mines of the United States. Together, this group, including the Potter slopes, formerly leased by the same corporation, in 1907 produced 1,403,745 long tons, or more than 51 per cent. of the total red ore for the district.

Practically all the ore produced in the district is manufactured into pig-iron in the vicinity of Birmingham. The ore is handled by 29 coke-furnaces, distributed as follows: In Birmingham city, eight furnaces, four of which belong to the Sloss-Sheffield Steel & Iron Co., one to the Tennessee Coal, Iron & Railroad Co., two to the Birmingham Coal & Iron Co., and one to the Williamson Iron Co.; at Ensley there are six stacks, and at Bessemer five stacks, of the Tennessee Coal, Iron & Railroad Co.; at Thomas, three stacks of the Republic Iron & Steel Co.; at Woodward, three stacks of the Woodward Iron Co.; and at Oxmoor, two stacks of the Tennessee Coal, Iron & Railroad Co. On the outskirts of the district are the furnace of the Southern Steel Co., at Trussville, and that of the Central Iron & Coal Co., at Holt, near Tuscaloosa.

In general, the furnaces run on a burden of coke, red ore, brown ore, and dolomite or limestone, though certain of them at times use only a self-fluxing red ore. The ores of the dis-

trict contain too much phosphorus to be converted into steel by the Bessemer process, but such pig-irons are being very successfully used for basic open-hearth steel making. The basic open-hearth process is employed by the Tennessee Coal, Iron & Railroad Co. at the Ensley rail-mill, which consists, including improvements now under way,[13] of one 15-gross-ton and two 20-gross-ton acid Bessemer converters, four new 60-gross-ton basic open-hearth tilting-furnaces, and two 65-gross-ton furnaces of the same type, ten 50-gross-ton open-hearth tilting-furnaces of an older type, and one 50-gross-ton stationary furnace, together with one coal reheating-furnace, soaking-pits, blooming-mill, rail-mill, and finishing-department, shops, and auxiliaries. The entire output of the company's six coke-furnaces is transferred as hot metal to the steel-mill, where it is made into billets and rails. The Ensley mills are adjacent to the Pratt coal-field and thus occupy a peculiarly advantageous location, since most of the ore and fluxing-materials are mined within a distance of from 3 to 10 miles.

Besides the Ensley steel-plant, this company operates at Birmingham steel-works equipped with one 20- and one 25-gross-ton basic open-hearth steel furnace, and at Bessemer, rolling-mills having a capacity of 60,000 tons annually of bars, plates, and light rails. The Southern Steel Co. also operates rolling-mills at Ensley.

In NE. Alabama, the Alabama Consolidated Coal & Iron Co. operates two coke-furnaces at Gadsden. The Southern Steel Co. has one stack and a basic open-hearth steel-plant at Gadsden. At Battelle one furnace ran for about two years on local red ore. There is one charcoal-furnace at Attalla, and one at Round mountain. The latter has been out of blast most of the time for several years.

---

[13] *Directory of American Iron and Steel Association*, 17th ed. (1908).

The Clinton Iron-Ore Deposits in Stone Valley, Huntingdon County, Pa.

BY J. J. RUTLEDGE, PH.D., BALTIMORE, MD.

(Chattanooga Meeting, October, 1908.)

I. Description of the Clinton Ores and Associated Rocks.

THE Clinton rocks in Stone valley comprise (1) thick layers of deep-red shale, (2) layers of reddish-gray shale interspersed with beds of sandstone and thin beds of extremely fossiliferous limestone, and (3) yellowish-gray shales alternating with thin layers of olive-colored shales. These layers of shale are all very thin and are at no point massive.[1]

A bed of gray-white sandstone varying in thickness from 12 to 30 ft. occurs in the upper portions of the Clinton shales. This is called the ore sandstone, a name given to it by Rogers and adopted also by the Second Pennsylvania Geological Survey. It is the surest guide to the presence of the iron-ore bed, which lies below its lower edges at distances varying from 10 to 20 ft. This ore sandstone, in the valley, lies generally at very low inclinations, but often is locally subjected to considerable changes in dip. At its outcrop this ore sandstone weathers very easily and yields a rather sparse crop of stone, composed of dirty, iron-stained boulders. Upon the roads these boulders are quite well rounded, and bear fragments of crinoid stems and other fossil forms.

The ore sandstone exhibits at many points evidences of former movements in this very great thickness of the shales, and as it is the only stratum in this series competent to transmit and record stresses due to structural movements, its evidence is of considerable importance in determining the occurrence of movements in times past.

---

[1] I. C. White. The Geology of Huntingdon County, *Second Geological Survey of Pennsylvania, Report of Progress T3*, p. 239 (1885).

1. *Occurrence of the Ore in Brush Ridge.*

A generalized section of the ore-bed and the rocks immediately above and below it in Brush ridge is practically:

	Feet.
Soil, x feet, usually varies from 5 to 10 ft.	x
Loosely-layered sandstone becoming gray in color,	8 to 20
White sandstone,	3 to 4
Brown sandstone, about	3
"Dirt-vein," from	3 to 4
Brown sandstone, "main roof" of the miners,	3
"Working ground,"	2 to 3
This is taken down in brushing the gangways and hence is called "working ground."	
Ore-bed, averaging 12 in. thick, often as thin as 6 in. and sometimes as thick as 18 in.	
Soft slate, buff in color, of unknown thickness. About 4 ft. of it is taken up in the gangway.	

A section of ore in the first left branch, Hunter drift, Brush ridge, is:

	Inches.
Soil,	x
Slate,	6
Lean ore not saved,	4 to 6
Slate,	8
Slate,	2
Ore, generally used when good quality,	2
Yellowish-green slate,	2
Soft, rich iron ore,	12
Soft bottom slate, grayish to yellowish in color,	x

A section of the ore-bed in Benson drift, at the west end of Brush ridge, is:

	Feet.
Sandstone, brown colored,	12 to 15
Rather hard, yellowish slates,	7 to 9
Soft yellow slate, "working ground,"	1.5
Soft, rich ore,	1.0
Bottom slates, yellow and olive-colored, with dirt-veins,	3.0

A. *The Ore-Stratum.*—The outcrop at the mill-dam presents the best, and perhaps the only, opportunity of observing the ore-bearing stratum in its original unchanged condition. Here it is a bed of very fossiliferous limestone about 8 or 10 in. thick. The fossils are remains of brachiopods, crinoids, bryozoans, and other organisms. In thin sections the lean ore shows the same organic remains, which will be referred to later. In

color the fossiliferous limestone is a bluish-gray when fresh and unweathered. The weathered edges of the outcropping limestone are dark-brown in color, and small cavities, evidently left by the solution and removal of the fossils, are seen. The weathered portion is of a rather lower specific gravity than the unweathered rock. In the bluish-gray unweathered limestone masses of what is apparently ferrous carbonate can be observed, and a portion of the mass surrounding the supposed siderite is seen to be calcite. This is observed both megascopically and microscopically.

Although the limestone at the mill-dam is one mass of fossil remains, these seem to be confined to bryozoans, brachiopods, crinoids, or smaller forms, and no corals were observed at this point.

Tracing the bed of fossiliferous limestone westward along the ridge, it is found to pass first, at a distance of about half a mile from the mill-dam, into thin, rather lean ore, of little commercial value, and finally into a soft, rich, typical oölitic ore at a distance of about three-fourths of a mile from the mill-dam. The first drifts into the east side of the ridge are at a distance of about half a mile westward from the dam, and from this point to the extreme western end of the ridge (or the Benson drift), waste-heaps from abandoned working-drifts are found every few hundred feet. At most of the drifts the larger portion of the material on the waste-heaps is composed of soft shales, greatly weathered, together with small pieces of soft, rich ore, but at two drifts, viz., the Hufford drift and the Parker, or road drift, about two miles west from the dam, there is a considerable quantity of hard ore lying on the waste-heap. Masses of hard ore are also found in the Bookhammer drift, at present in operation, which produces soft ore as well. This hard ore is similar in appearance to the soft ore, that is, it is oölitic, but is apparently cemented by silica and contains remains of corals not found in the soft ore. These corals are *heliolites* and are composed of calcium carbonate; in addition to the limestone corals, which are clearly defined against the hard ore and are not at all replaced by the iron, masses of gray-blue shale appear in the hard ore.

As the ore is followed in its course from the Parker drift westward its color changes from brick-red to a dull black at

the Mule drift. This latter ore is reputed to be the richest ore dug on the ridge. The ore in the Bookhammer and Benson drifts, a short distance westward from the Mule drift, is of the ordinary brick-red color characteristic of the other drifts.

All through the ridge, but more especially towards the western end, the ore-bed is found to be much contorted. The average dip is about 10° to 15°, but rises to as much as 55° in some places, and is often overturned.

Jointing, both primary and secondary, is observed in the ore-bed. These joints are always found full of water when first opened. Vugs filled with clay are also found near the joints. Below the ore-bed lies a dark olive-colored shale. Apparently the slope of the ridge over the ore-bed has considerable influence on the richness or leanness of the ore. Over the Mule drift the surface-slope is quite gradual, being about 5°. The miners report that, as a rule, soft ore occurs under the ravines and hard ore under the rolls. They also affirm that if, in drifting across the measures to reach the ore-bed, shale or slate is met with before reaching the ore sandstone, the ore sandstone is hard, unbroken, and unweathered, and the ore below the sandstone is also hard. Where the ore sandstone lies under soil only and is broken, the ore below is soft and rich. The ore is hard and lean under the minor rolls on the slope of the ridge and soft under the ravines. The hard ore occurs as points or triangular-shaped bodies with their pointed ends pointing up the slope of the ridge.

Ground-water is found in the slates and shales under the hard ore, while it occurs in the shales and slates over the soft ore in the same bed. This change occurs within very short distances.

The ore sandstone seems to retain and carry the water and, apparently, has an intimate relation with the richness of the ore. There is considerable ground-water in all the mines, and most of the abandoned drifts, where good, soft ore was found and worked, have issuing from them small streams of water of excellent quality. Aside from the occurrence at the mill-dam no outcrops of either soft or hard ore in place are found on Brush ridge, though elsewhere in Stone valley several such exposures of soft ore are found. Occasionally masses of iron pyrites are found in the center of masses of solid, massive

hematite, but the aggregate amount of such finds is not large. In the Bookhammer drift the "working ground" overlying the ore-bed is found to be of fair quality, though hard and unsuited for reduction in a charcoal cold-blast furnace. Old miners report such occurrences common in earlier workings on the ridge.

B. *Topographical Conditions.*—Brush ridge is a low and rather narrow monoclinal, between 2 and 3 miles long, composed of Clinton strata. At no point throughout its entire length does the ridge rise to any considerable height, being usually at an elevation of from 200 to 500 ft. above the drainage-level of the region. It has undergone considerable erosion and the surface is well-rounded. Numerous ravines cut the north side, which is composed of the outcropping edges of the shales, but the southern slope is quite gradual, varying from 10° to 15°, and is cut by very few ravines and these few are minor ones. No good exposures of the rocks are seen at any point along the southern slope except near the mill-dam at the eastern end of the ridge. The mines are all at or near the western end of the ridge, at which point the soil is brick-red to yellow in color, soft and rather sandy.

At the mill-dam the ore-bed lies nearly horizontal, with a dip to the SW. of not more than 4°, which increases to between 10° and 15° within a quarter of a mile west from the mill-dam. There seems to be a definite relationship between the angle of inclination at which the iron-ore bed lies, the slope of the overlying surface, and the richness of the ore. For example, the ore is lean and contains a large percentage of $CaCO_3$ at the mill-dam, where it lies nearly horizontal and is covered by heavy dark-gray unweathered shale, which lies at a dip of about 20°. As the ridge road is followed westward from the mill-dam the soil changes in color from dark gray to a buff color and rounded, worm-eaten pieces of ore sandstone appear in it. The ore sandstone is broken, lies near the surface of the ground, and is not covered by shale. Under such conditions the iron-ore bed is generally found soft and rich. Moreover, at all points where the ore-bed outcrops in other portions of Stone valley between Standing Stone and Tussey mountains, the surface-material over the rich, soft ore-beds always lies at a slope of between 10° and 20°.

Here and there on the southern slope are small minor ravines which carry the drainage in times of storm. Under most of these ravines, as far as the mine-workings have penetrated, there is generally found soft, rich ore. Under the minor rolls lying between these small ravines carrying wet-weather streams, the ore is, as a rule, hard, siliceous, and lean.

The county-road leading from Greenwood furnace to Huntingdon extends along the foot of the eastern slope of Brush ridge and defines almost exactly the line denoting the boundary between the soft, rich, fossil ore lying above it and the hard, lean, unaltered ore lying below it. In all the cross-cuts driven to reach the ore-bed in Brush ridge it has been found that this is prevailingly the case.

As the road is followed south from Greenwood furnace the ridge is seen to decrease in height, and the slope of the eastern side is more gradual and the outlines are more rounded.

2. *Physical Characters of the Clinton Ore in Stone Valley.*

The Clinton iron-ore in Stone valley is very similar to the typical oölitic Clinton ore found in New York and other States, and often called fossil ore. It can be divided roughly into two varieties, hard ore and soft ore. The former variety is not used here in the manufacture of iron, as, owing to its refractory nature, it is unadapted to smelting in the charcoal cold-blast furnaces in Stone valley; it is also mined with difficulty, carries considerable silica, and uniformly occurs under the hogbacks, or small local rolls, in the side of Brush ridge. Many organic remains are found in this ore, such as corals, brachiopods, and crinoid stems. Scales of micaceous hematite also occur in it.

The soft ore is composed of small oölites, can be rather cheaply mined, and is easily reduced in the charcoal-furnace. If unweathered, the ore is hard, calcareous, and lean, and contains many fossil remains; but if soft, few organic remains are discernible in it. Jointing is quite well shown in both the hard and the soft ore.

The ore can be found in all varieties, grading from the gray-blue limestone, containing but a small percentage of iron and composed almost entirely of organic remains (chiefly brachiopod shells) through the hard, lean, and siliceous ore to the soft, rich,

granular ore. The oölitic character is prominent in both the hard and the soft ores, but the brachiopod remains, so numerous in the limestone at the outcrop, have nearly all disappeared from the hard ore, and are rarely observed in the specimens of soft ore. At the mill-dam the outcrop of the ore is merely a bluish-gray limestone, made up almost entirely of remains of shells and crinoid stems.

The soft ore when freshly mined is quite soft and is easily dug with the pick, but hardens somewhat when exposed to the air. The hard ore is mined only with difficulty. Both varieties of ore occur in the same bed, and they grade into each other and seem to bear important relations to the topography and drainage of the surface.

A. *Oölitic Structure.*—Like all other occurrences of Clinton iron-ore, that of Stone valley is oölitic, the grains being about 1 mm. in size, and of various shapes, some being spherical, others oval, and still others having their longitudinal dimension much greater than their cross dimensions. The oölitic structure is plainly visible megascopically in all varieties, both hard and soft, except in the lean ore at the mill-dam. Beyond the mill-dam the fossils disappear almost entirely, and the ore, both in hard and soft varieties, is truly oölitic, though brachiopod remains occur occasionally in the hard ore and rarely in the soft ore.

The ovules, when freshly mined, are brownish-red in color and of earthy appearance, but in the hard variety the oölites on drying harden and become glazed, and have a reddish-brown color, resembling very much in shape and appearance ordinary flaxseed. These oölites have their long dimension approximately horizontal, and cause the ore-masses to cleave easily in that direction.

B. *Concretions.*—The mass of the ore, both hard and soft, is composed of concretions of a diameter of about 1 mm. These concretions are apparently made up of coatings of iron oxide about a center of some hard material, evidently organic in character and sometimes composed of calcite. They are of various shapes, some much greater in length than in width, others oval in form, and still others circular in cross-section. The nucleus is, in some cases, calcareous, bearing coatings of iron oxide on its outer surface, and between the concretions the

cementing material is also calcareous in character and evidently calcite, as the interference-colors with crossed nicols and the cleavage-lines seem to indicate this mineral. Relatively, only a portion of the bed is ore-bearing. Under the microscope, by far the larger portion of the field is occupied by the calcite forming the filling between the concretions, the iron coatings on the concretions being a minor constituent in the field.

Cross-sections of these concretions yield, in thin sections, a remarkably regular structure.

Sections for microscopic examination were cut from specimens of both hard and soft ore, but the former yielded much better results than the latter, as, owing to the nature of the soft ore, it was very difficult to secure thin sections. Although the greater portion of the sections examined came from the hard ore, sections were cut from hand-specimens showing the limestone in all stages of the transition from fossiliferous, lean, gray-blue limestone to the soft, rich ore. In all the rock- and ore-sections examined the concretions were found in varying quantities. They were recognized in the limestone at the outcrop near the mill-dam, in the hard ore, where the coating of iron oxide rendered their outlines conspicuous, and in the soft ore, though here their outlines were, to a certain extent, masked by the soft ore in the grinding.

These concretions are associated with other forms, probably bryozoans, and are undoubtedly organic in character. About 4 g. of the soft ore was treated with dilute hydrochloric acid in a Gooch crucible for one week, and the residue, after being well dried in a desiccator, was mounted in Canada balsam. The resulting material, apparently siliceous, seemed to be mostly amorphous in character, but occasional bodies, possessing the characteristics of concretions, were found after the material was mounted. These concretions gave interference-colors of quartz and an axial cross under the microscope, and were made up of successive coatings of silica, which was stained with iron oxide. It was not possible to use a high power upon the concretions, as, owing to their very delicate structure, they could only be mounted with extreme difficulty and the results yielded thick sections. A very slight contact with the needle-point sufficed to cause the kernel to separate from the outer coatings or caused a separation between the individual coatings. Even

a very little pressure upon the cover-glass resulted in the breaking-up of the concretions. All that were successfully mounted, however, exhibited the concentric structure and the axial cross with crossed nicols. The silica seemed, in every case examined, to be stained with iron. Not more than four successive coatings were recognized in any case, but this does not prove that there were not additional coatings present, as it was only possible to use a low-power objective for the above-given reasons. In nearly every case the concretions were oval in shape, though occasional ones possessing circular cross-sections were observed. The oval shape may have resulted from structural movements incidental to causes of dehydration of these ores. Samples of both hard and soft ore yielded concretions, though they were much more abundant in the siliceous residue resulting from treatment of the former variety with hydrochloric acid than in that from the latter. When the siliceous residue was immersed in turpentine and placed under the microscope, the concretions could be plainly seen with a low power. With every slight movement of the watch-glass the concretions turned over, revealing the fact that they were very thin in respect to their length and width. They appeared nearly as thin, looking at them edgewise and parallel to their long dimension, as they would appear if taken from a microscopic rock-section. These iron-stained siliceous oölites, usually of oval shape, have a very glassy appearance with plane polarized light, but with crossed nicols they show their concentric character. They are a relatively minor constituent, judging from the amount yielded by treating both varieties of the ore with hydrochloric acid.

3. *Chemical Character of the Clinton Ore of Stone Valley.*

The most striking result brought to light by the analysis of the two samples of soft and hard ore, given later, is the very large amount of calcium carbonate contained in the latter as compared with the soft ore. This result is quite in accordance with the conditions at all mines where hard ore was found. Masses of blue-gray limestone and corals were found in all the hard-ore occurrences.

It will be noticed that there is a very great difference in the amount of insoluble siliceous material in the two varieties of

ore, the hard ore having but 3.58 per cent., while the soft ore contains 16.40 per cent. The siliceous concretions contained iron oxide stains even after having been treated with hydrochloric acid for some time. This would seem to indicate that the iron and silica of these concretions were deposited together, as claimed by Professor Smyth[2]; and that they are so much more numerous in the soft ore than in the hard ore, where the unchanged limestone occurs, indicates that the solutions carrying silica and iron effected the replacement of the limestone of the ore-bed. The gray and olive-colored bands of varying thickness, scattered through the Clinton shales, which owe their color probably to the presence of ferrous silicates, may have yielded a portion of the iron. The following analyses of ore and limestone are of interest:

Bulk Sample of Soft Ore Taken from All Operating Mines on Brush Ridge, Huntingdon County, Pa.

Specific gravity, 4.424.

	Per Cent.
Ferric oxide (Fe_2O_3),	68.35
Alumina (Al_2O_3),	5.33
Calcium oxide (CaO),	0.16
Magnesium oxide (MgO),	0.04
Manganous oxide (MnO),	0.31
Potassium oxide (K_2O),	0.48
Phosphoric acid (P_2O_5),	0.52
Sulphuric acid (SO_3),	none
Carbonic acid (CO_2),	0.16
Water and organic,	8.15
Insoluble siliceous material,	16.40
	99.90

Insoluble siliceous material.

Insoluble silica (SiO_2),	12.03
Soluble silica (SiO_2),	3.15
Ferrous oxide (FeO),	0.16
Alumina (Al_2O_3),	1.06
	16.40

Hard Ore, Parker Drift, Brush Ridge.

Specific gravity, 3.495.

	Per Cent.
Ferric oxide (Fe_2O_3),	48.06
Ferrous oxide (FeO),	1.44
Alumina (Al_2O_3),	4.02
Calcium oxide (CaO),	22.06

[2] On the Clinton Iron Ore, *American Journal of Science*, Third Series, vol. xliii., No. 258, pp. 487 to 496 (June, 1892).

	Per Cent.
Magnesium oxide (MgO),	0.56
Manganous oxide (MnO),	0.17
Potassium oxide (K_2O),	0.19
Phosphoric acid (P_2O_5),	1.04
Sulphuric acid (SO_3),	none
Carbonic acid (CO_2),	18.84
Insoluble siliceous material,	3.58
	99.96

Insoluble siliceous material.

Silica (SiO_2),	2.58
Ferrous oxide (FeO),	0.22
Alumina (Al_2O_3),	0.78
	3.58

Fossiliferous Limestone from Mill-Dam, Brush Ridge.

Specific gravity, 2.696.

	Per Cent.
Insoluble siliceous material,	15.88
Calcium oxide (CaO),	41.60
Magnesium oxide (MgO),	0.37
Ferrous oxide (FeO),	2.12
Ferric oxide (Fe_2O_3),	2.35
Alumina (Al_2O_3),	1.62
Phosphoric acid (P_2O_5),	0.17
Carbonic acid (CO_2),	34.16
Loss on ignition,	1.62
Undetermined,	0.11
	100.00

Insoluble siliceous material.

Silica (SiO_2),	12.44
Ferrous oxide (FeO),	0.48
Alumina (Al_2O_3),	2.96
	15.88

4. *Fossils.*

Organic remains are found in varying proportion all through the single bed of ore near Brush ridge. In the lean calcareous ore at the mill-dam they make up the entire mass of the rock, and brachiopod shells, portions of crinoid stems, and other organic remains are very abundant. As the bed becomes richer in ore, fossils are less abundant, until in the hard and soft varieties of ore, remains of organisms, except the oölites, are exceptional. Occasionally brachiopod shells replaced by hematite, and fossil casts composed of the same material, are found in the soft ore, but these occurrences are relatively rare. In the hard ore, however, organic remains are more numerous,

brachiopod shells and corals occurring in nearly every bed examined.

At one drift, viz., the Parker, or road drift, remains of corals, wholly composed of calcium carbonate, were found. These corals were very well preserved, and it was possible, after cutting sections from them, to determine the species. A few brachiopod remains, poorly preserved, were also found at this point. These corals all occur imbedded in hard ore and lie with their long dimension parallel to the bedding of the ore. Scales of micaceous hematite were also found associated with these coral remains at the Parker drift. Indeed, an examination of nearly every find of hard ore along Brush ridge yielded coral remains in an excellent state of preservation, but as the quantity of hard ore at the Parker drift was greater than at any other drift, this drift yielded the majority of the specimens of corals.

5. *Key-Rocks for Locating the Iron-Ore.*

A. *The Ore Sandstone.*—The ore sandstone in Brush ridge is usually very much weathered to brown-gray, considerably worm-eaten masses, which are solid at very few places. At one point, the Evans drift, about midway between the two extreme ends of the ridge, the ore sandstone is, however, found massive and extremely hard. Under this hard stone, however, no workable ore was found.

The ore sandstone varies in thickness from 12 to 30 ft. The distance driven in it, to reach the ore, by the drifts or cross-cuts, is about 80 ft. Towards its upper portion the ore sandstone, when massive, is in beds from 2 to 4 ft. thick, but its lower layers are always soft and iron-stained.

When weathered, the ore sandstone is shown to be traversed by thin iron-stained bands, a fraction of an inch in thickness, and running parallel to the bedding. This feature of the weathering is especially well-marked in the exposures at the Evans drift.

When bleached white by exposure to the sun and weather, the ore sandstone shows fossil casts, shells, and crinoid stems, and presents a porous, spongy appearance. Where the ore sandstone is overlain by shale, in place and unweathered, the

ore below the ore sandstone is hard and lean; on the other hand, when there is no unweathered shale overlying the ore sandstone, and the ore sandstone is broken by vertical jointings (as it is at nearly every place near the western end of the ridge), the ore below is soft and rich.

B. *The Clinton Lower Shales.*—The shale overlying the ore sandstone gradually changes in character beyond the milldam. There it is hard, dark-gray in color, and has undergone very little weathering and the soil is of a dirty-gray color. But farther along the ridge the shale is soft, yellow to buff in color, and apparently has been very much weathered, and the soil is buff to brick-red in color. Percolating waters have penetrated to all portions of the mass of shale lying over the ore, and the freshly-mined shale has an unctuous feel when held between the fingers.

Scattered along the county-road are many worked-out drifts with their old waste-heaps. At all these openings the shale presents the porous, worm-eaten appearance indicative of the action of percolating waters. At most of the drifts the ore sandstone is covered only by the soil resulting from the weathering of the overlying shale, and the ore sandstone boulders are at the surface. At one point, the Evans drift, however, the shale was found overlying the ore sandstone to a depth of about 10 ft. The Evans drift was driven in a roll between two minor ravines carrying wet-weather streams. It is situated about midway between the two extreme ends of the ridge, on its eastern slope. The ore sandstone was here found to be unbroken under the roll (though broken ground and ore sandstone boulders appeared about 200 ft. to the west of it) and was extremely hard. So difficult was the work of drilling this rock that the drift was abandoned. Some lean ore was found in the ore sandstone, which was banded parallel to the bedding, but no soft ore was ever found in this drift. The miners report that wherever the shale is found overlying the ore sandstone the ore below the latter is hard, indicating a connection between the weathering of the shale and the richness of the ore.

II. The Origin of the Clinton Iron-Ores.

1. *Previous Interpretations.*

There is very little literature dealing with the occurrence and mode of origin of the Clinton ores. Several of the State Geological Surveys have noted the presence of Clinton iron-ores in their respective States and have made reference to them in their reports, but, on the whole, very little careful scientific work in regard to the origin of these ores has been performed. However, the question is a very interesting one and has lately engaged the attention of several careful workers. In the following pages a *résumé* of the most important articles is given.

The Clinton iron-ores are clearly bedded deposits, and must, as is commonly accepted, owe their richness of iron-content to four causes: (1), to an original deposition of the iron, at the time the present ore-beds were laid down; (2), to a removal of the soluble constituents of the present ore-bed—probably calcium carbonate—and a resultant relative enrichment of the ore-bed, the iron being left by the solution which carried away the calcium carbonate; (3), to the addition of iron from extraneous sources and an absolute enrichment, or (4), to a combination of two or all of these processes.

The following brief summary of the views held by the quoted authorities, shows that they may be clearly grouped in three categories, viz.: (1) According to Newberry and Chamberlin, the ore-formation is due to original deposition of the iron-content. (2) According to Shaler, Russell (in part), and Foerste, the ore-formation is the result of secondary enrichment. (3) According to Smyth, the iron-content is due to original deposition and a subsequent relative enrichment by the removal of the calcium carbonate.

Prof. H. D. Rogers.[3] Clinton ores originated like other sedimentary rocks, their iron-content being derived from the waste of surrounding rocks and laid down at the same time the ore-bearing bed was deposited. There was also an additional deposition of iron from the overlying shales and a relative

[3] *Proceedings of the Boston Society of Natural History*, vol. vi., pp. 340, 341 (1858). Iron Ores of the Surgent [Clinton] Series, *Geology of Pennsylvania*, vol. ii., p. 729 (1858).

increase in the iron-content of the bed, due to weathering-action.

Prof. N. S. Shaler.[4] The ores were not included in the present iron-ore beds at the time of their deposition, as conditions varied so much at different points that this would have been impossible. The ore-occurrences are due to replacement of limestone-beds by iron-bearing solutions derived from overlying shales. The iron could not have been deposited as far from shore as the limestones were.

Prof. J. S. Newberry[5] declared that the Clinton deposits were marine and not marsh deposits. The iron was derived from the drainage of the rocks to the northeast of the Clinton sea, and deposited as are the Swedish-lakes ores, *i. e.*, it is an original deposition. While in transportation the iron was soluble protoxide, but became insoluble by oxidation, and was laid down as limonite, which later became dehydrated. Crinoid fragments served as nuclei for the concretions. The large amount of phosphorus in the ore indicated organic origin.

Prof. T. C. Chamberlin[6] found no marine fossils in the Wisconsin deposits of Clinton age and accepted Newberry's theory of the origin as similar to that of the "mustard-seed" ore of the Swedish lakes.

He concluded that the oölites arose from successive concretions of iron oxide about quartz-grains as a nuclei.

Prof. I. C. Russell[7] asserted that the main factor in the formation of Clinton iron-ore beds was the influence of weathering, and thought this was indicated by the appearance and chemical composition of the ores at various horizons. Where unweathered the ore was thicker than where weathered.

Dr. Aug. F. Foerste[8] brought out the fact that the oölitic grains did not manifest the spherical character of oölites of other geological age, and that they were probably bryozoan fragments. Iron oxide has replaced the bryozoan fragments to

[4] Notes on the Investigations of the Kentucky Geological Survey During the Years 1873, 1874, and 1875, in *Reports of the Kentucky Geological Survey*, Second Series, vol. iii., pt. iii., p. 163 (1877).

[5] *Geological Survey of Ohio, Geology*, vol. iii., pp. 5–7 (1878).

[6] *Geology of Wisconsin*, 1873-1879, vol. i., p. 179 (1883).

[7] Subaërial Decay of Rocks and Origin of the Red Color of Certain Formations, *Bulletin* No. 52, *U. S. Geological Survey*, p. 22 (1889).

[8] On the Clinton Oolitic Iron Ores, *American Journal of Science*, Third Series, vol. xli., No. 241, pp. 28 to 29 (Jan., 1891).

FIG. 1.—HARD ORE. MAGNIFIED ABOUT 40 DIAMETERS.

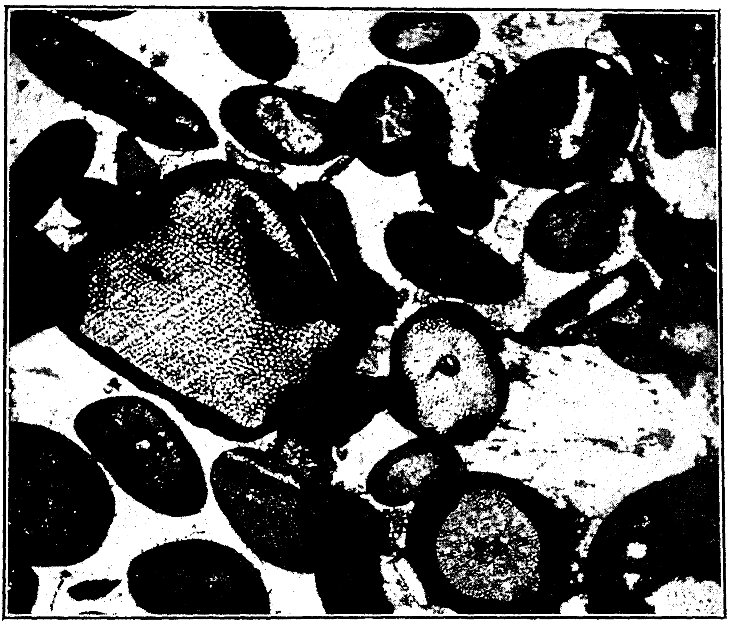


FIG. 2.—HARD ORE, PARKER DRIFT. IRON CONCRETIONS HAVING AS NUCLEII BROKEN CRINOID FRAGMENTS. MAGNIFIED ABOUT 40 DIAMETERS.

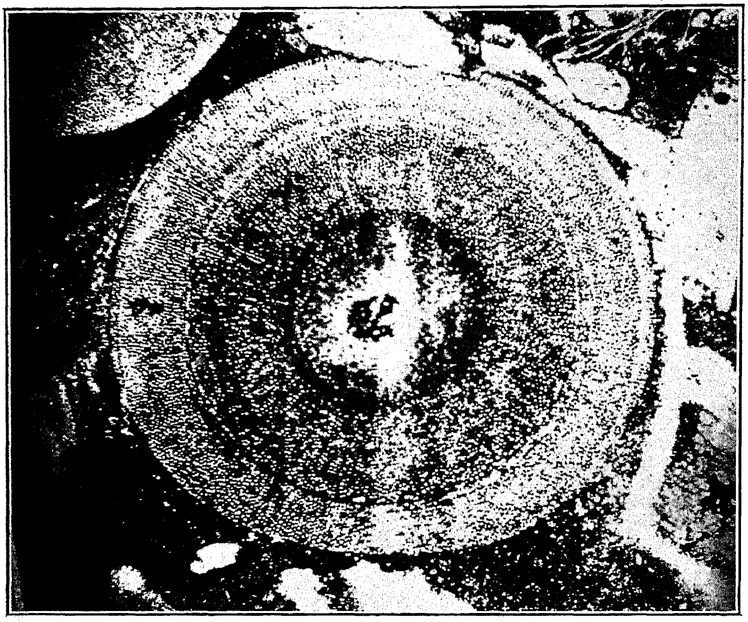


FIG. 3.—HARD ORE. CRINOID PLATES. MATERIAL OF DISK IS CALCITE. MAGNIFIED ABOUT 40 DIAMETERS.

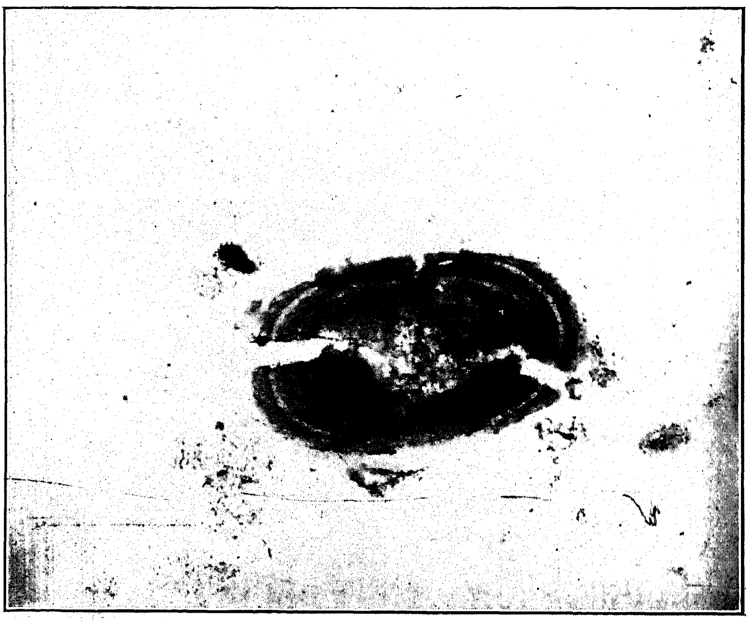


FIG. 4.—SILICEOUS CONCRETIONS. MAGNIFIED 40 DIAMETERS.

a greater or less degree, as well as the cementing material between the various fragments. He did not think that the oölitic character was due to concretionary segregation of iron oxide, but to the gradual replacement of bryozoan fragments.

Prof. C. H. Smyth, Jr.,[9] thought that the application of the replacement theory to the Clinton iron-ore was not warranted by the facts in the case. The iron could not have passed through calcareous rocks without being precipitated by them, yet the ore occurred, in some places, overlain by limestones and calcareous shales. He found that the concretions in the lean ore are as ferruginous as those in the rich ore. Accordingly the substitution of the fossil fragments took place previous to their consolidation into a bed by the cementing material. If the ore is a replaced limestone, ferrous carbonate should be found, and yet it has never been discovered. The iron did not come from above, for Clinton ore-beds are often found lying horizontal with no chance for the action of percolating waters. There was an original deposition of ore. No iron has been added to the bed since its deposition; there has simply been a removal of the calcium carbonate and a relative enrichment of the bed in iron. The iron is secondary with respect to the organic fragments, but is primary with respect to the ore-bed. In some localities the weathering-action has been more important than at others. He concluded that the iron oxide and silica had been deposited together from solution in meteoric waters. The iron was retained in such waters by organic matter. There is an intimate connection between silicic acid and organic matter which leads to combinations of silicic acid, iron, and organic acids in soils and a deposition of the iron and silica together. The oölites were not originally calcareous.

2. *Genetic Phenomena of Stone Valley Ores.*

An examination of the Stone valley occurrences seems to indicate that the ore-beds must occur under certain topographic conditions in order to insure an iron-content sufficient

---

[9] Die Hämatite von Clinton in den östlichen Vereinigten Staaten, *Zeitschrift für praktische Geologie*, vol. ii., pp. 304 to 313 (Aug., 1894), and On the Clinton Iron Ore, *American Journal of Science*, Third Series, vol. xliii., No. 258, pp. 487 to 496 (June, 1892).

to make them of economic importance. Throughout the valley they lie upon the sides of small ridges, the outcrop of the beds usually lying close along the strike of the ridge in conditions favorable to an easy and slow movement of meteoric waters. Moreover, the richer ore-beds generally lie at about the same angle of dip as do the sloping sides of the ridge above them, and a rich bed has usually a dip as great as 10° or 15° or greater.

The second important factor is weathering. At all the occurrences the action of the atmospheric and meteoric waters is noticeable. Where the topographic requisites are fulfilled weathering results, and the appearance of the buff- and red-colored clays resulting from the weathering of the shales indicates to the prospector the likelihood of soft ores below. Another result of arrested weathering is the occurrence of hard ore in the minor rolls, between the small ravines which carry wet-weather streams. The extremely hard ore sandstone at the Evans drift and the hard ore beneath it show that the requisite weathering-action has not occurred at this point.

The presence of ground-water below the hard ore and above the soft ore indicates the important part that such water plays in the leaching of these beds.

That the present ore-bed is a replaced limestone is shown by the fossiliferous limestone-bed at the mill-dam, which can be traced until it becomes the hard or soft ore-bed. The limestone among the hard ore at the Parker drift also indicates that the present ore-bearing bed was once a limestone, as do also the corals found in the hard ore which are not entirely replaced by iron-ore.

The dark-blue fossiliferous limestone probably owes its color to the presence of ferrous iron carbonate, for small particles can be observed in the hand-specimen which have the cleavage of siderite and are either crystals of ferrous carbonate or calcite crystals carrying more or less iron oxide. Probably the iron carbonate is in isomorphous combination with the calcite. The fossiliferous limestone contains but about 2.12 per cent. of ferrous oxide, FeO, and 2.35 per cent. of ferric oxide, Fe_2O_3, so that there must have been some addition of iron. There is very much more calcium carbonate in the hard ore than in the soft ore, indicating that it has not been removed by waters

bearing iron, while the soft ore contains more silica than the hard ore, probably denoting that the waters which dissolved and carried away the calcium carbonate from the limestone left iron and silica behind.

The fossils observed in the fossiliferous limestone and in the hard ore are identical, except that those in the latter have been largely replaced, with the single exception of the corals, which were never found to have been replaced. The soft ore contains the same crinoid fragments as are found in the fossiliferous limestone and in the hard ore, and these fragments are sometimes wholly replaced by silica. An examination of the fragments of crinoids, bryozoans, and brachiopods found in all the ore-occurrences reveals the fact that they can be traced in all stages of their replacement by iron oxide, from the fossiliferous limestone, where they are generally entirely calcareous, through the hard ore, where they are partly replaced, to the soft ore, where they are less numerous, but still plainly recognizable and nearly entirely replaced by iron oxide. The crinoid fragments are the most numerous of all the organic fragments, and are generally the only remains found in the soft ore.

The concretions, either of round or of oval shape, are siliceous, and may be isolated by treatment of the fossiliferous hard or soft ore with hydrochloric acid. They have a concentric character, and the silica is nearly always iron-stained. They give an axial cross with crossed nicols. As compared with the crinoid fragments they are a minor constituent of the ores and are very much smaller. They are extremely fragile, and can be mounted only with great difficulty.

An examination of the sections cut from the fossiliferous limestone and the hard and soft ores demonstrates that the original cementing material between the organic fragments and the concretions is calcite. This mineral constitutes what may be called the "ground-mass" of the rock- or ore-sections. Replacement appears to commence first on this calcite cement, and by means of the cleavage-cracks has access to the organic fragments. Nothing was noticed which would lead to the inference that the fragments had been replaced before they were cemented together, except that there was an apparent slight replacement of the interior of the fragments in some of the limestone-sections where there was no external coating of iron

oxide. The analogous replacement of Medina sandstone indicates that these Clinton shales in Stone valley contain sufficient iron to produce beds of iron-ore when it is concentrated within the space of a few inches, as it is in the Clinton bed in this valley. An examination of sections cut from the Medina sandstone and from the ore shows a filling of iron oxide between the quartz-grains of the sandstone. An analogous change occurs in the limestone, except that the quartz-grains are here represented by calcareous organic fragments, while the filling between is calcite. These shales are competent iron-forming beds.

The conclusion that the iron-content of the Clinton iron-ore beds of Stone valley is due mostly to replacement by removal and enrichment, seems unavoidable when it is considered that but a portion of the fossiliferous limestone or of the hard ore is found to contain iron oxide when examined in thin sections under the microscope. The calcite cement makes up by far the greater portion of the section. An analysis of the limestone shows that it contains but 2.12 per cent. of FeO and 2.35 per cent. of Fe_2O_3. This seems much too small an iron-content to yield as rich an ore as the soft ore, simply by removal of the calcium carbonate. Field-conditions, such as the occurrence of weathered shales, buff-colored clays, and iron-stained sandstones, prove that the action of replacement is still progressing.

3. *Origin of the Clinton Iron-Ore of Stone Valley.*

As a result of my studies, I have reached the conclusion that but a very small part of the iron-content found in the present ore-bed was deposited in it when the bed was laid down. This small amount of iron is represented by the iron-content of the siliceous concretions. By far the greater portion of the present iron-content of these Stone Valley ore-beds is believed to have been laid down in the beds of shale overlying the ore-beds as an original constituent of such shale-beds, and subsequently transferred from these shale-beds to the bed of fossiliferous limestone which formerly occupied the location of the present ore-bed. I agree with Professor Smyth as to the original deposition of the iron-content of these beds only so far as the siliceous concretions are concerned, but I differ with him in ascrib-

ing the present richness of the beds very largely to the addition of iron and not alone to the removal of the limestone, as Professor Smyth asserts. The grounds for these conclusions, briefly summarized, are:

A. The character of the iron in the ore-concretions where it is associated with silica;

B. The invariable association of the soft, rich ore with the leached, decolorized shales, and of the hard, lean ores with unweathered bright-red shales;

C. The relations of the ores to the shattered sandstones and to the topographic situation of the ores;

D. The fact that analogous replacements are now taking place in the Medina;

E. The observed progressive steps in the transformation of the limestone to an ore, which may be followed in the field, in thin sections under the microscope, and in chemical analyses; and, finally,

F. The absence of conditions, such as local crumpling, including a shrinking of the strata, pointing to a relative rather than an absolute enrichment of the ores.

A. *The Iron of the Concretions.*—Upon treating the fossiliferous limestones and the hard and soft ores with hydrochloric acid, small concretions are isolated. These concretions are composed of silica and iron oxide combined, and appear to be made up of concentric coatings of these substances. This is the case with the concretions in the limestone as well as those in the hard and soft ores. On the other hand, the organic fragments, composed of remains of crinoids, bryozoans, and brachiopods, are entirely calcareous in the limestone, and are gradually replaced by iron oxide until they are almost entirely composed of iron oxide in the soft ore. That the iron and silica probably entered the limestone together is shown by the fact that the crinoid fragments are often not affected by the hydrochloric acid, and can be isolated in the same manner as the siliceous concretions, since their iron-content is bound up with the silica.

B. *Association with Shales.*—Every occurrence of the richer iron-ores shows them to be in association with altered shales, whose weathered character is indicated by the great predominance of the yellow and gray colors which they possess. These shales are brick-red in the upper portion, or that part overlying

the iron-ore beds, and inclined to be massive in places; under the ore-bed they are brown to gray in color, and in the lowermost portions contain numerous olive-colored bands, in which the color is due to the presence of ferrous silicates.

It is evident from a study of the conditions in Stone valley that weathering-influences and rock-disintegration have been very important factors in the formation of the Clinton ore of this region. At all the mines where soft ore is found the shales immediately above the ore-bed are found to be of a light-buff color, to have an unctuous feel and to show the effects of percolating waters. These shales have been deprived of their iron-content by these waters, which have then carried it to the underlying limestone and there deposited it as an ore. That this must be the case can be shown by a consideration of the columnar section of the Clinton group in Brush ridge. At the top of the columnar section is a mass of brick-red shale which locally becomes quite massive. This red shale is succeeded by beds of dark-gray shale which lie just above the ore sandstone. In both these masses of shale there are found, here and there, olive-colored bands, varying from 1 to 20 ft. thick, which owe their color to the presence of ferrous iron silicates. That portion of the column lying below the ore-bed and above the Medina contact contains numerous beds of olive-colored shales, but no limestones are found in these beds near them. In other portions of central Pennsylvania the Block iron-ores, which are really ferruginous sandstones, occur at this horizon. The Upper Clinton shales, dark-red in color, where unweathered, and containing the olive-colored beds, have probably furnished the source for the Clinton ore of this region.

C. *Relations to Overlying Sandstone and to Topography.*—Everywhere in Stone valley where Clinton ore is found the shales over the ore have been eroded and the ore sandstone appears at the surface, broken and weathered. On the other hand, on Standing Stone mountain, where the topographic conditions were such that the Upper Clinton shales were but slightly eroded, no soft ore is found. The ore must have come from the Upper Clinton shales. In this connection it is interesting to refer to a paper by James P. Kimball,[10] in which he speaks of the oc-

[10] Genesis of Iron-Ores by Isomorphous and Pseudomorphous Replacement of Limestone, etc., *American Geologist*, vol. viii., No. 6, p. 356 (Dec., 1891).

currence of Clinton iron-ore on the flanks of Tussey, Dunnings, and Wills mountains in southern Pennsylvania, in these words:

"All of these ores owe their development, as I believe, exclusively to secular replacement of elevated parts of these limestones [thin crinoidal limestones]—not, as sometimes explained, to direct sedimentation in whole or in part. For wherever oolitic iron-ores are developed within the Clinton series, they are found to graduate into non-ferriferous limestones, more or less crinoidal, and usually in circumstances only moderately favorable to weathering action. An equally significant fact is the absence of valuable iron-ores where the Clinton limestone, as in southern Ohio, is massive and unaccompanied by a considerable thickness of overlying shales. Wherever, on the other hand, the limestone occurs in numerous thin beds, and so alternates with more or less ferruginous shales; or again, wherever overtopped by shales, it seldom fails, especially in steep dips, to graduate unequally into oolitic hematite by replacement."

The opinion that beds of iron-ore are derived from shales and slates does not seem unwarranted when some of the facts surrounding the occurrence of different iron-ores are considered. All through the geological column from Cambrian ores to those of Carboniferous age, iron-ores are found near shales and slates, the latter generally pyritous. It is only necessary that there be soluble limestones present to receive the ore and topographic conditions favorable to a slow movement of percolating waters to insure ore-deposition. Organic matter is usually abundant in the shales. The Appalachian ore-occurrences are in areas where great structural movements have occurred, and the beds carrying the ore have been left in a position favorable to a slow and efficient action of drainage-waters. The fact that the maximum thickness of Clinton iron-ores, about 22 ft., occurs in the southern Appalachians, where little more than 200 ft. of the entire Clinton group remains, seems significant.

There seems to be an intimate relationship between the amount of shattering which the ore sandstone covering the ore has received and the weathering of the limestone. The fossiliferous limestone exposed at the mill-dam, which is here of a blue-gray color, and evidently contains ferrous carbonate, lies practically horizontal under a rather thickly-bedded mass of dark-colored, unweathered shale about 30 ft. thick. This bed of fossiliferous limestone is one mass of fragments of brachiopods, bryozoans, and crinoids, and can be identified by these organic remains. Following the ridge westward, the shale over the fossiliferous limestone disappears and exposes the ore sandstone,

which lies immediately below the shale and over the ore. The ore sandstone is found to be broken soon after leaving the mill-dam, and the soil in the roadway changes from a dark-gray color to a light buff. Boulders of the ore sandstone are here much shattered and weathered and show, on freshly-broken surfaces, traces of hematite. All through Stone valley, at every occurrence of soft ore it is found to be overlain by broken ore sandstone. Another argument in favor of the importance of weathering-effects in the formation of these Clinton iron-ore beds in Stone valley is the fact that the beds are generally rich and soft when lying at a dip of from 10° to 20°, and under a surface where the slope of the covering strata is about the same.

D. *Analogous Replacements.*—That these Clinton shales contain disseminated in them sufficient iron to furnish the 12- to 18-in. bed of oölitic iron-ore now found in Stone valley can be demonstrated by a reference to a very lean and highly siliceous iron-ore found at various points in Stone valley, viz., at Steffy's hotel, at Joseph Bowman's house, near Old Monroe furnace, at the place where the county-road crosses Big Laurel run, on Greenlee mountain, and at E. Musser's house. At all these places the white Medina sandstone projects through the Clinton lower shales, nearly always forming an arch, with the crown partly eroded. In all such occurrences the shale is found to be weathered to a soft clay, of a buff color; and boulders of the white Medina sandstone, as well as the beds in place, are found to be transformed to a lean iron-ore which yields on analysis from 17 to 30 per cent. of iron and is very high in silica. The sandstone can be found in all stages of transition, from the pure white sandstone through that bearing a yellowish-brown outer shell of iron oxide about ⅛ in. thick with a clear white interior, to the fine-grained ore, which is an entirely transformed sandstone. The soil yielded by the shale appears to act most rapidly and efficiently on the sandstone boulders, but also affects the sandstone when in place. Microscopic sections of the sandstone in all processes of transformation, show it to be of a rather fine grain and exhibit the iron oxide filling all the spaces between the separate grains of the sandstone. That this sandstone ore was not an original deposition can be shown at Steffy's hotel, where it occurs as an ore immediately in front of the hotel; but 300 ft. to the north,

where the sandstone is not covered by disintegrating shale, it is of a pure white color and contains no iron oxide. On the sides of the road-cutting near the hotel small pieces of sandstone, containing about 20 per cent. of iron oxide, can be observed projecting from the bed of loose, dark-gray shale in which they lie. It is thus seen that analogous replacements are taking place to-day, except that the cementing substances in sandstones and not fossiliferous limestones are being substituted.

E. *Observed Progression in Alteration of Limestone to Ore.*—That the ore-bed was once a limestone is shown by the occurrence at the mill-dam and by the finding of limestone-masses in the Parker drift and the Bookhammer drift. The limestone in both of the last-named places is of a bluish-gray color and contains coral remains in an excellent state of preservation. Moreover, the "working ground," *i. e.*, the lean sandstone overlying the ore, is at Bookhammer drift almost rich enough in iron to be classed as an ore, thus indicating that there has been a replacement of this sandstone at this point. That the mine-water is not found above the hard ore, but below it, and is found above the soft ore, would seem to indicate the importance of such waters in the present instance. In central Pennsylvania the prospectors for brown hematite (or limonite) expect to find it only in places where there is limestone, hence it would seem reasonable to expect a replacement of the limestone in this instance. The unchanged masses of limestone are usually found on the crests of minor rolls on the sides of the ridge, indicating that their presence is due to inefficient drainage-action. Another important fact bearing upon the question of the action of meteoric waters is that the hard ore is usually massive and difficult to mine, and is not broken up by numerous joints and vugs as is the soft ore. The joints and cavities, existing generally only in the soft ore, contain clay and water when in place and when freshly opened up. Hard ore becomes extremely hard when exposed to the sun and atmospheric influences, while soft ore disintegrates under like conditions. Soft ore when freshly mined contains a considerable amount of water between its individual grains, while hard ore does not, and never disintegrates upon exposure to the atmosphere.

Megascopically, the individual grains of the hard ore are

roughly oval in shape and rather flat. In thin sections of both hard and soft ore, as well as in those of the bluish-gray fossiliferous limestone, numerous remains of brachiopods, bryozoans, and crinoids were found. The latter fragments are most persistent and are clearly visible in the soft ore. A large portion of the crinoid fragments appear to be replaced by iron and silica, as the iron-stain was found in the interior of the crinoid fragments even under a high power. Under the microscope, the crinoid fragments appear concentric, and a replacement of their calcite by silica and iron oxide might give rise to a simulated concentric or oölitic structure. The replaced fragments have a concentric appearance, but are truly of organic character. Iron oxide has replaced the substance of these organic fragments from the exterior, approaching them by means of the cleavage-cracks in the calcite cement lying between the different organic fragments. Some organic fragments in the limestone and lean ore are wholly calcareous. There seems to be a change in the cement first, and, succeeding this, a replacement of the organic fragments. In thin sections of soft ore the cement is wholly replaced, while the crinoid fragments, some rounded and others oval-shaped, are seen rather dimly but still distinctly. When these fragments of crinoids have been replaced by silica they become isolated in treating the soft ore with hydrochloric acid, in the same way as the concretions are separated.

An examination of the results of the chemical analyses of the hard and soft ores and the fossiliferous limestone seems to yield results which bear out the theory of replacement and enrichment. Little or no calcium carbonate is found in the soft ore, while there is more than 22 per cent. of calcium oxide and more than 18 per cent. of carbon dioxide in the hard ore, in which the pure, unchanged limestone-masses are found, which seems to point to the lack of action of percolating waters. It should be noticed that there is a considerable difference in the amount of insoluble silica found in the two varieties of ore, the soft ore having 16.40 while the hard ore has but 3.58 per cent.; the iron and silica probably, in large measure, entered the limestone together.

F. *Relative vs. Absolute Enrichment.*—There seems to be a lack of conditions indicating a relative rather than an absolute en-

richment. The overlying shales show the action of percolating water and a probable removal of their iron-content. An examination of the ore-sections and of the results of chemical analysis of the ores and limestone seems to develop the fact that the original iron-content of the limestone is too small to furnish such a rich ore as the Stone valley occurrence merely by the results of relative enrichment.

III. Original Source of the Iron.

The main point at issue between Professor Smyth and me is the original source of the iron. Professor Smyth holds that all of the iron was originally deposited in the limestone, the $CaCO_3$ being later removed and the iron-content of the bed thereby relatively increased. I hold that the iron was only in part original, and that the calcium carbonate was not simply removed but really replaced by iron derived from the overlying shales through the agency of percolating waters.

If Professor Smyth's hypothesis is correct, there should be:

1. A sufficient amount of iron in the original limestone to make an ore.

2. A distinct shrinkage of the ore-bed as compared with the thickness of the unaltered limestone or an unusually porous ore.

3. This shrinkage should be accompanied by evidences of more or less local movement consequent upon the attempt of the adjacent rocks to adapt themselves to the shrinkage, and this should be observed both in microscopic sections of the ores and rocks and in the field-occurrences.

4. The specific gravities of the ores and rocks should also indicate the changes produced by the leaching-out of the calcium carbonate.

A comparison of the results of the analyses of the fossiliferous limestone and the hard and soft ores yielded important results, which appear to refute completely the hypothesis of original deposition. According to the values obtained, 1 cu. ft. of the fossiliferous limestone contains about one-fourteenth as much iron as the hard or soft ores; and if there has been no addition of iron from extraneous sources, there must have been a removal of 13 cu. ft. of limestone for every cubic foot of ore now found in the bed. Such excessive shrinkage is not demanded by the field-conditions, but, on the contrary, field-

observations and detailed microscopic studies show that no such shrinkage could have taken place.

The ore-beds traced along their strike from the richest concentrations to the unaltered fossiliferous limestones show no appreciable change in thickness, and the contiguous beds show no local adjustments or crinklings, such as must have occurred if the ore is only the ferruginous remnant of the original limestone. Microscopic examinations point to the same end, in that the rocks and ores examined show no minute adjustments or rearrangements resulting from a loss of the calcareous portions, but on the contrary the progressive steps in the gradual replacement of the calcium carbonate by iron from some outside source may be easily traced in the different sections.

If leaching was the only factor brought into play, and the enrichment was only a relative one and not absolute, then the following conclusions may be drawn: In the fossiliferous limestone the $CaCO_3$ and SiO_2 equal 90 per cent. of the whole rock, while in the hard ore the sum of the $CaCO_3$ and SiO_2 equals 43 per cent. of the whole, therefore the 10 per cent. of the limestone which is other than $CaCO_3$ and SiO_2 is concentrated to 57 per cent. in the hard ore, or as 1 : 5.7. This being true, the sum of both irons of the limestone is increased in the ratio of 1 to 5.7, which equals $4.47 \times 5.7 = 25.48$ per cent. But the iron of the hard ore is 49.50 per cent., or double what concentration alone would produce, therefore there must have been an addition of iron from exterior sources in order to account for the iron-content of the soft ore.

Looking at the matter from the standpoint of the relative specific gravities of the fossiliferous limestone and the hard and soft ores, we have the following conclusions: Taking into consideration the fact that field-conditions show no evidences of shrinkage of the ore-beds (and that they are hard and compact and not at all porous), then if leaching alone acted and no matter was added from outside, and the stratum of limestone did not shrink (and field-conditions show no evidences of shrinkage), then the hard and soft ores should be quite porous and much lighter than the limestone, as they, composed of the non-leachable matter, must occupy the space originally occupied by the limestone. This is contrary to the actual conditions, as both the hard and soft ores are compact and heavier than the lime-

stone; hence, there must have been an addition of iron from the outside. There is 90 per cent. of extractive or soluble matter, *i. e.*, $CaCO_3$ and SiO_2, and 10 per cent. of non-extractive or insoluble matter. This 10 per cent. of non-extractive material becomes 57 per cent., or is increased in the ratio of 1 : 5.7. This being true, and no material being added from extraneous sources, the specific gravity of the hard ore should be the following, using the determined specific gravity of the limestone, 2.69:

$1:5.7::x:2.69$; whence $x = 0.47$, which is, of course, absurd.

1. *Conclusions.*

The observations made in the field while studying the Stone Valley ores, as well as the study of the microscopic sections of the ores and limestones, lead me to the conclusion that by far the greater part of the iron of these ores is due to the replacement of limestone and concentration of the ore. The very small amount of iron represented by the iron oxide intimately combined with the silica in the concretions isolated by treating both varieties of ore with hydrochloric acid, may be due to original deposition. It is also probable that all of the iron present as the carbonate in isomorphous mixture with the calcium carbonate was deposited originally, although some of this may have arisen in the early replacement of the lime of the carbonate by the iron. But the quantity of originally-deposited iron so found is extremely small in comparison with the total iron-content of the ore. The iron oxide found in the calcite cement and forming coatings around the organic fragments represents by far the greater amount of iron in the ore.

In the Clinton upper red shales the iron exists as a peroxide, and in this form is insoluble and cannot be affected by ordinary surface-waters, but rain-water containing carbon dioxide falls upon this shale after coming into contact with the organic matter in the leaves and soil; chemical reactions result, the peroxide (ferric oxide) is reduced to ferrous oxide, and carbonic acid formed by decomposition of organic matter unites with the ferrous oxide to form iron carbonate or siderite. This is carried in solution until it comes in contact with the fossiliferous limestone, when the lime is replaced by iron and the calcium carbonate is carried away in solution. The iron is prob-

ably present first in the limestone in isomorphous combination, as the appearances seem to indicate this.

Professor Smyth failed to find any siderite in his rocks, and concluded that the iron-ore must have been an original deposit and not formed as the result of enrichment through the intermediate formation of a carbonate. I have, however, found what is, apparently, siderite in my rocks, and the presence of the iron in combination with carbon dioxide is corroborated by the chemical analyses. If one makes a distribution of the different elements of the analysis of limestone among the minerals found, assuming that part of the lime is united with the phosphorus to form a lime phosphate, it readily appears that the soluble ferrous iron is present in the form of siderite unless part of the CO_2 is unsatisfied. The same is true in part for the hard ore, but does not hold in the case of the soft ore, where all of the iron is oxidized to the ferric state.

The iron carbonate was formed under conditions in which organic matter was in excess; later percolating waters acted, after structural movements had occurred, and the water, now containing small quantities of organic matter, caused a deposition of ferric oxide and left the ore as it is found to-day, soft and pure.

The Clinton Iron-Ore Deposits in New York State.

BY D. H. NEWLAND, ALBANY, N. Y.

(Chattanooga Meeting, October, 1908.)

DURING the year 1907 an investigation of the Clinton formation in New York has been carried out under the direction of the State Geologist, and a full account of the results has been prepared for publication.[1] The more important economic features of the report are summarized in the present paper.

Previous knowledge of this, the type, section of the Clinton has been contributed mostly by Hall and Vanuxem in their reports for the Natural History Survey of New York, which were issued in their final form in 1842–43. Brief accounts of the ore-occurrences have been given by B. T. Putnam,[2] J. C. Smock,[3] A. H. Chester,[4] and E. C. Eckel.[5] Prof. C. H. Smyth, Jr.,[6] has discussed the origin of the Clinton ores in two papers, which, so far as the local occurrences are concerned, have removed all doubt that may have existed on that subject.

Distribution of the Clinton Formation.

The Clinton strata in New York State are restricted to a single belt, which extends from the eastern central part to the Niagara river, which it crosses, and thence continues for some distance into the province of Ontario. The New York

---

[1] Newland, D. H., and Hartnagel, C. A. Iron Ores of the Clinton Formation in New York State, *Bulletin No.* 123, *New York State Museum* (1908).

[2] Notes on Samples of Iron-Ore Collected in New York, *10th Census of the United States*, vol. xv. (1880).

[3] First Report on the Iron Mines and Iron-Ore Districts in the State of New York, *Bulletin No.* 7, *New York State Museum* (1889).

[4] *The Iron Region of Central New York.* Printed for the Utica Mercantile and Manufacturing Association, Utica, N. Y. (1881).

[5] The Clinton Hematite, *Engineering and Mining Journal*, vol. lxxix., No. 19, p. 897 (May 11, 1905).

[6] On the Clinton Iron Ore, *American Journal of Science*, Third Series, vol. xliii., No. 258, pp. 487 to 496 (June, 1892). Also, Die Hämatite von Clinton in den östlichen Vereinigten Staaten, *Zeitschrift für praktische Geologie*, vol. ii., pp. 304 to 313 (Aug., 1894).

section has a length east and west of 225 miles. The outcropping edges of the strata spread over a width of 5 miles in the middle part of the belt, around Oneida lake and for some distance westward, but the width gradually diminishes away from that locality more rapidly towards the east, owing to the increasing inclination of the beds in that direction.

Until recently it was supposed that the Clinton formation was represented in southeastern New York, and certain strata found along the Shawangunk and Skunnemunk mountains have been mapped and described as such. Without entering upon details of the matter here, sufficient evidence has been forthcoming from re-examination of these localities to justify the statement that the Clinton formation does not appear to the east of Otsego county, where the belt above mentioned terminates by thinning out of the strata. This fact may be regarded as throwing a good deal of doubt upon the existence of Clinton rocks in eastern Pennsylvania, as described in the geological reports of that State. At any rate, such beds, if they exist, cannot be a continuation of the New York belt, though the latter shows close relation with the areas in central Pennsylvania where the Clinton is exposed along the Appalachian folds.

General Geology.

Stratigraphic Relations.—The New York section of the Silurian rocks, to which the Clinton formation belongs, is made up of the following elements in ascending order:

System.	*Group.*	*Formation.*
Silurian.	Cayugan, ...	Manlius limestone. Rondout waterlime. Cobleskill limestone. Salina beds.
	Niagaran, ...	Guelph dolomite. Lockport dolomite. Rochester shale. Clinton beds.
	Oswegan, ...	Medina sandstone, including Oswego sandstone and Oneida conglomerate.

The whole succession from the base to the top of the Silu-

rian is conformable, sedimentation having been continuous throughout the entire era.

The character of the sediments as a whole shows progressive change from the conglomerates and sandstones that make up the lowest members to finer sands and muds prevailing in the Clinton and Rochester formations, and lastly to the limestones and dolomites which predominate in the upper formations. The change is doubtless to be regarded as reflecting a certain amount of coastal depression, producing a gradual deepening of the waters with the progress of time, but it is not supposed that the shales and limestones required anything more than moderate depths for their accumulation. Many of the limestones are plainly fragmental in character, composed of broken and comminuted fossils that appear to have been worked over and ground by the waters before consolidation. Moreover, the shales of the Rochester and Clinton formations show abundant evidences of the prevalence of beach and shoal-water conditions by the ripple-marks, shrinkage-cracks, and tracks of invertebrates, just as are found on present sea-beaches.

The widespread occurrence of hematite-layers in the Clinton, which, it is believed, has its explanation in the prevailing physical features of that time, is comparable with the salt- and gypsum-deposits which are so extensively developed in the Salina formation. The ore-beds, thus, do not exemplify extraordinary or isolated conditions, but rather those common to Silurian times in eastern North America. The occurrence of salt-springs in the Medina may also be considered in this connection.

General Structure.—The Clinton beds throughout the State have a low southerly dip, in conformity with the slope of the original coastal plain on which they were laid down. Their uplift from sea-level must have taken place very gently, and they have since remained practically undisturbed. When regarded in their entirety, however, they do show a certain amount of bending or warping, incident to their elevation, the effect of which is to give the area a broad and very shallow synclinal arrangement, the relatively depressed portion occupying the central part. The eastern section of the belt has a dip to the west of south, while on the western section the dip is slightly east of south. On the extreme east, in Herkimer county, the outcrop lies at about 1,400 ft. A. T. At Clinton it is 700 ft.

In Wayne county, near the middle, the ore-bed lying close to the base of the formation outcrops at about the level of Lake Ontario, 346 ft. A. T. West of there the elevation increases more slowly, and at Rochester the base of the formation is about 420 ft. A. T. Farther west the syncline appears to be interrupted by a minor anticlinal undulation, so that in Niagara county the beds have a southwesterly dip and their base is found at 400 ft. A. T.

The dips are uniform for long distances. They range from 40 to 50 ft. to the mile in Wayne and Cayuga counties, to 80 ft., or slightly more, on the western end, and to 150 ft. and more on the eastern section. Throughout the middle part they are scarcely greater, probably, than the slope of the sea-bottom on which the strata were accumulated.

Character of the Clinton Succession—There is much variation in the character, thickness and faunal contents of the Clinton beds, which, on that account, well deserve the name Protean, first assigned to them by Vanuxem.

In Niagara county, limestone is the main member, with a thin layer of shale, but no hematite so far as has been observed. In Monroe county the limestone becomes subordinate to the shale, while a thin bed of iron-ore appears in the lower section. Frequent changes with respect to the relative development of the limestones and shales are the rule throughout the part of the belt included in Wayne and Cayuga counties. In Wayne county a second ore-horizon is found above the main bed, and this appears frequently as far east as Oneida county. East of Cayuga county, the limestones are scarcely recognizable, becoming shaly and lacking the usual fossil character. In Oneida county thin-bedded sandstones are prominent and often gain preponderance over the shales. Another feature in this part is the appearance of the red-flux bed, a low-grade fossil hematite, from 4 to 6 ft. thick.

The stratigraphic succession along the belt will be shown by a few sections that have been prepared from field-observations and records of drill-borings. The sections follow in order from west to east.

1. *Section at Niagara Falls.*

	Ft.
Limestone, fossiliferous, ...	12
Limestone, compact, few fossils, ...	14
Shales, green, ...	6

2. *Section at Rochester.*

	Ft.	In.
Limestone, fossiliferous,	18	
Shale, green, carrying graptolites,	24	
Limestone (Pentamerus),	14	
Iron-ore, fossil hematite,	1	2
Shale, green,	24	...

3. *Part Section at Ontario, Wayne County.*

	Ft.	In.
Shale, calcareous.	2+	
Limestone (Pentamerus),	8	
Iron-ore, fossil,	1	10
Shale, green, calcareous,	10+	...

4. *Section at Wolcott, Wayne County.*

	Ft.
Limestone, shaly,	13
Shale, dark, with graptolites,	44
Ore, fossil,	1
Limestone (Pentamerus),	22
Shale,	62
Limestone, shaly (lower Pentamerus),	13
Ore, fossil,	2
Shale,	2+

5. *Part Section in Cayuga County, near Red Creek.*

	Ft.	In.
Shale, calcareous, graptolite layers,	77+	
Limestone, shaly,	17	
Shale,	66	4
Ore, fossil,	...	4
Limestone, gray fossiliferous,	2	...
Shale, purple,	5	...
Limestone, gray fossiliferous,	1	4
Ore,	2	6
Shale,	3	6

6. *Part Section at Brewerton, Onondaga County.*

	Ft.	In.
Shale, olive, with 2 in. fossil hematite,	124+	...
Ore, oölitic,	1	4
Sandstone with shale-layers,	5	...

7. *Section at Lakeport, Madison County.*

	Ft.	In.
Limestone, shaly, with 6 in. fossil hematite,	17	...
Shale with limy band,	227	...
Ore, mixed with limestone and shale,	2	6
Shale, calcareous in places,	45	...
Ore, oölitic,	1	...
Sandstone, shaly at top,	2	...

8. *Section at Clinton, Oneida County.*[7]

	Ft.
Sandstone, calcareous, thin shale-layer,	50+
Ore, red flux,	6
Sandstone, calcareous,	6
Shale, blue, thin sandstone-layers,	15
Ore, oölitic,	2
Shale,	2
Ore, oölitic,	1
Shale, blue, thin sandstone-layers,	100+

The section at Lakeport shows a total thickness of 295 ft., which is about the maximum for the Clinton in New York State. The sections at Ontario, Red creek and Brewerton do not extend through the entire formation.

From Clinton eastward the bounds of the formation are rather indefinite. A detailed study of the beds is needed to establish the relations between the Clinton and Rochester formations in this part.

Occurrence of Iron-Ore.

The hematite-seams attain their fullest development in the stretch of 125 miles from eastern Oneida to western Wayne county. At Rochester, the extreme westerly point where the ore is known to be represented, there is a single bed of fossil hematite 14 in. thick. This is very likely a continuation of the seam which extends across Wayne county and is mined around Ontario, 15 miles northeast of Rochester. On the east end the ore-seams thin out rapidly after passing the Oneida-Herkimer county line, and the entire formation comes to an end near the eastern border of Herkimer county.

From the results of recent test-drilling and field-observations, it appears that the ore is mainly gathered into a few areas which succeed each other along the outcrop after intervals characterized by thin seams or by their almost complete absence.

The area centering about Clinton has been most extensively mined of all. There are two seams here, an upper, of fossil character, called the red-flux bed, and a lower oölitic bed that is sometimes split into two portions by a layer of shale or sandstone. The fossil ore is too lean to be used in the furnace,

[7] By C. H. Smyth, Jr., in Kemp's *Ore Deposits of the United States*, 3d ed., p. 116 (1900).

containing little more than 20 per cent. of iron. It has a thickness of 6 ft. The oölitic bed can be traced for more than 10 miles along the outcrop. It ranges from 20 to 36 in. thick, and averages 40 per cent., or a little more, in iron. In the town of Verona, Oneida county, there is a subordinate area underlain by a fossil bed, measuring from 12 to 20 in., which was worked in the early days.

At Brewerton, on the west end of Oneida lake, a test-hole penetrated a bed of oölitic hematite 16 in. thick. This does not appear anywhere in outcrop, and its extent and volume are as yet unknown. An analysis showed 34 per cent. of iron.

The next area is encountered in northern Cayuga county, beginning near Sterling station and extending westward into Wayne county. The mine-excavations at Sterling station show from 30 to 36 in. of fossil ore, while a mile south along the dip the bed has been found with a thickness of more than 40 in. The bed has been proved as far west as Wolcott, where the drill encountered 21 in., while an overlying 12-in. seam comes in at that place. A drill-hole put down at Red creek, midway between Wolcott and Sterling station, showed the main seam to be 30 in. thick. The average iron-content of the bed may be placed at from 35 to 38 per cent.

The continuation of the ore immediately west from Wolcott has not been prospected. It may be assumed, however, that the main seam thins in that direction or is broken up by intercalations of limestone, which is in accordance with the results found at Wallington, 10 miles from Wolcott. At about the same distance beyond Wallington, in the town of Ontario, Wayne county, is an area that contains a bed of fossil ore from 18 to 30 in. thick. This seam has been worked along the outcrop for a distance of 5 or 6 miles, and explored by the mining companies for several miles farther. It appears to be the same bed that is represented in the section at Rochester. The ore from the open-cut mines in the town of Ontario averages a little more than 40 per cent. of iron.

The ore-seams that are of minable grade are thus quite thin as compared with some of the Southern Clinton deposits. The maximum thickness is probably 4 ft. in New York State. The seams show, however, surprising persistence, maintaining their thickness scarcely undiminished for miles along the strike and,

so far as examined, also on the dip. They are, of course, lenticular in their general shape, but the lenses are much more flattened and elongated than would appear to be usual with the other Clinton strata.

The amount of ore represented in the New York belt when subjected to estimate reaches very large proportions. Sufficient data have not been secured up to the present to afford a close calculation of the whole amount, and anyway but a small part can be considered as minable under conditions that may obtain within a long period of time. As indicative in some measure of the resources, an estimate of the quantity available in the three principal areas, after the elimination of seams less than 18 in. thick and exceeding 500 ft. from the surface, shows a total in round numbers of 600,000,000 tons. Taking 2 ft. as the minimum thickness to be considered, the estimate would be very largely reduced.

The greater part of the resources within easy reach from the surface is contained in the western areas of Cayuga and Wayne counties. The inclination of the beds in this section is not more than 50 ft. to the mile, generally, while the surface of the country has a very moderate rise southward; as a consequence, the ore-seams could be followed for 5 or 6 miles back from the outcrop before the limit of 500 ft. depth would be reached. Test-holes put down at distances of between 3 and 4 miles from the outcrop have encountered the ore with its average thickness.

In Oneida county the beds have a dip of about 150 ft. to the mile, and they reach the 500-ft. limit usually within 2 miles from the outcrop.

Mining-Operations on the Clinton Belt.

The Clinton ore has been mined in New York State since the beginning of the last century. There is record of a mining-lease that was granted in Oneida county as far back as 1797, and a small quantity of ore was shipped from Wayne county during the war of 1812. At the time of the Natural History Survey of the State (1836–41), charcoal-furnaces and forges were run on Clinton ores in Wayne, Madison, and Oneida counties.

Mining, so far, has been conducted on a small scale, and

operations have been wholly or partly suspended during times of severe market-depression. The aggregate output to date may be placed at from 4,000,000 to 5,000,000 tons. For the last few years the average annual output has been about 75,000 tons. The production of 109,025 tons in 1907 is perhaps nearly as large as that for any previous year.

The mines that have been recently active are those of the Franklin Iron Manufacturing Co., and C. A. Borst, at Clinton; Fair Haven Iron Co., at Sterling station, Cayuga county; and the Furnaceville Iron Co., and Ontario Iron Co., near Ontario, Wayne county.

In Oneida county openings have been made along the ore-outcrops at intervals all the way from the Oneida-Herkimer county line to Verona station on the west. The method of stripping, once used exclusively, has now given way to underground mining. The advancing long-wall system is employed in both mines at Clinton, and it seems to be well adapted to the conditions there, as well as to other parts of the Clinton belt. By taking advantage of the surface-features, it has been possible at Clinton to follow the ore from the outcrop, and to make use of the slight inclination of the beds to secure natural drainage. The main entries or gangways are run in an easterly or northeasterly direction across the dip. From these, branches turn off at every 100 ft. to the working-face. The ore measures 30 in. on the average, and about 2 ft. of the overlying shale is taken down to afford working-room. The shale is packed behind the face, and the roof further secured by placing wooden posts temporarily in front of the pack. The posts are removed, so far as possible, after the roof is supported, to be used again in the same manner. Owing to the soft character of the overlying beds, the roof settles readily and uniformly, with little or no danger from falls. The lower part of the ore-seam is removed first by drilling a line of holes diagonally from near the top. After these are blasted, the upper part with 2 ft. of shale is taken out by drilling horizontal holes. The ore is trammed by hand or mules to the loading-stations outside.

The composition of the oölitic ore as mined is shown in Table I.

TABLE I.—*Analysis of Clinton Oölitic Iron-Ores.*

	1.	2.	3.	4.	5.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Metallic iron (Fe).............	48.42	30.08	56.37	44.10	50.68
Silica (SiO_2).....................	11.57	29.72	9.98	12.63	11.34
Alumina (Al_2O_3)........	3.92	4.13	2.40	5.45	3.91
Manganous oxide (MnO).....	0.19	0.37	tr.	0.15	1.63
Lime (CaO).....................	5.80	8.57	1.54	6.20	3.97
Magnesia (MgO)................	2.27	1.96	0.30	2.77	2.21
Sulphur (S)......................	0.28	0.837	Nil	0.23	Nil
Phosphorus (P).................	0.754	0.67	0.541	0.65	0.915

NOTE—1 is by C. H. Smyth, Jr. 2 relates to the bottom layer of oölitic ore from Franklin mine, J. B. Britton, analyst. 3 is of ore from same mine, probably upper or main layer, J. B. Britton, analyst. 4 is an average analysis of ore from the Franklin and Clinton mines (Franklin Iron Mfg. Co.), made in 1873 by A. H. Chester. 5 is of ore from Clinton mine, J. B. Britton, analyst.

The ore mined by the Franklin Iron Manufacturing Co. has been used in its furnace near Clinton. Most of the product made by C. A. Borst is sold for grinding into metallic paint and mortar-colors. This ore is shipped with an average of 45 per cent. of Fe. The median 12 in. or so of the oölitic bed will run above 50 per cent. of Fe.

The property of the Fair Haven Iron Co. is situated just south of Sterling station, between the Lehigh Valley and the New York Central (R., W. & O. branch) railroad-lines. The company was organized in 1906. Thus far a few thousand tons of ore have been taken out by trenching with a steam-shovel. The trench begins about 400 yd. southwest of the station and follows the line of the outcrop to the east. The fossil bed is found below 10 to 25 ft. of soil and rock. The loose overburden represented by soil and glacial materials varies from 1.5 to 10 ft. The covering is loosened by drilling 6-in. holes down to the ore with an oil-well rig and blasting. The shovel, mounted on a track laid in the trench, loads the loosened material into cars, which run out at the east end and are dumped to the north. The ore is taken out by hand after drilling and blasting. It is then loaded into cars, which enter the trench by means of a spur from the main lines.

The ore-seam varies from 30 to 38 in. thick, the average minable thickness being probably 30 in. It is inclosed directly by from 1.5 to 2 ft. of limestone above and 5 ft. of shaly limestone below. The shipments from the property have

averaged between 36 and 38 per cent. of Fe. An analysis of a sample has been supplied by W. L. Cumings: Fe, 34.98; SiO_2, 6.01; Al_2O_3, 0.95; MnO, 0.47; CaO, 13.96; MgO, 7.80; S, 0.044; P, 0.351; CO_2, 19.39; H_2O and organic matter, 0.45.

The Furnaceville Iron Co., with properties north of Ontario Center, has perfected to a high degree of efficiency a method of open-cut excavation. The ore-lands owned by the company extend for more than 4 miles in an east and west direction. The plan of work consists, briefly, in opening longitudinal trenches, the first along the northern limits of the property near the ore-outcrop, and the following ones in parallel order progressively with the removal of the ore from the preceding trench. At the present time about 20 ft. of overburden is taken off, while in the first cut, some 40 rods to the north, the overlying soil and rock was 6 ft. thick. The trenching is done by a single 100-ton shovel, which excavates a width of 60 ft. The shovel loads into the skips of a hoist, a portable incline structure provided with two skips, each of 6 cu. yd. capacity, that carry the rock up to the top of the spoil-bank. Following this is a smaller shovel for loading the ore, which is hoisted by a derrick on to cars for shipment. A spur from the R., W. & O. railroad extends along the trench on the side opposite the spoil-bank, and is moved back from time to time with the advance of operations towards the south. The overburden at the present stage consists of 10 ft. of limestone, somewhat shaly towards the top, and about the same thickness of soil and glacial materials. It is loosened for the shovel by drilling and blasting. The 6-in. holes, made by churn-drills, extend downward to the ore-bed, and into the latter for about 3 in. They are placed 16 ft. apart, the first row being 6 ft. from the edge of the trench. A layer of limestone, from 15 to 18 in. thick, remains on the ore and has to be taken off by hand. The ore is broken by powder, after holes 3 ft. apart, extending a few inches into the underlying shale, have been made by steam-drills. The ore is shipped to Emporium, Pa. Its composition is illustrated by the following analysis, reported by W. L. Cumings: Fe, 44.12; SiO_2, 11.74; Al_2O_3, 0.48; MnO, tr.; CaO, 7.34; MgO, 3.76; S, 0.028; P, 0.494. Figs. 1, 2, and 3 show the mining-methods as conducted by the Furnaceville Iron Co. at Ontario, N. Y.

The open-cut or trenching method of work is undoubtedly

the most economical where the overburden does not exceed 20 ft. or so on the average. With a 2-ft. ore-seam, which will yield approximately 8,000 gross tons to the acre of surface, the cost of stripping and removing the ore under ordinary conditions, with from 15 to 20 ft. of overburden, may be placed at about $1.50 a ton. With a 3-ft. seam, the cost should not exceed $1.25 a ton. Underground-mining has been carried on at Clinton for a little less that $1.50 a ton, though I am not certain that this cost includes a charge for preliminary development.

The lack of progress which has characterized the past history of mining along the Clinton belt may be attributed largely to the low iron-content of the ore. With only from 35 to 45 per cent. of Fe, the ore will not stand transportation for any great distance, though the cost at the mine may be relatively low. The presence of from 10 to 20 per cent. of lime and magnesia carbonate in the ore is a valuable feature, especially for mixture with high-grade material like the magnetite-concentrates from the Adirondacks, which are of siliceous character. There is no doubt that furnaces situated within easy shipping-distances of the two districts would command a permanent and relatively cheap ore-supply.

Origin of the Ores.

The Clinton hematites, so far as they occur in New York State, are to be considered, unequivocally, sedimentary deposits, laid down in water along with the accompanying shales, limestones, and sandstones. This view, originally set forth by James Hall, in his report for the Natural History Survey, has been discussed by C. H. Smyth, Jr.,[8] with such detail as to leave little to be added.

It is true, of course, that the fossil ores are replacements in the sense that there has been a chemical interchange of iron for the lime of the fossils, but this process was active during the period of sedimentation and not afterwards. Circulations of ground-waters subsequent to the emergence of the strata above sea-level have had no place in the accumulation of the ores.

---

[8] *Zeitschrift für praktische Geologie*, vol. ii., pp. 304 to 313 (Aug., 1894). A briefer discussion was published in *American Journal of Science*, Third Series, vol. xliii., No. 258, pp. 487 to 496 (June, 1892).

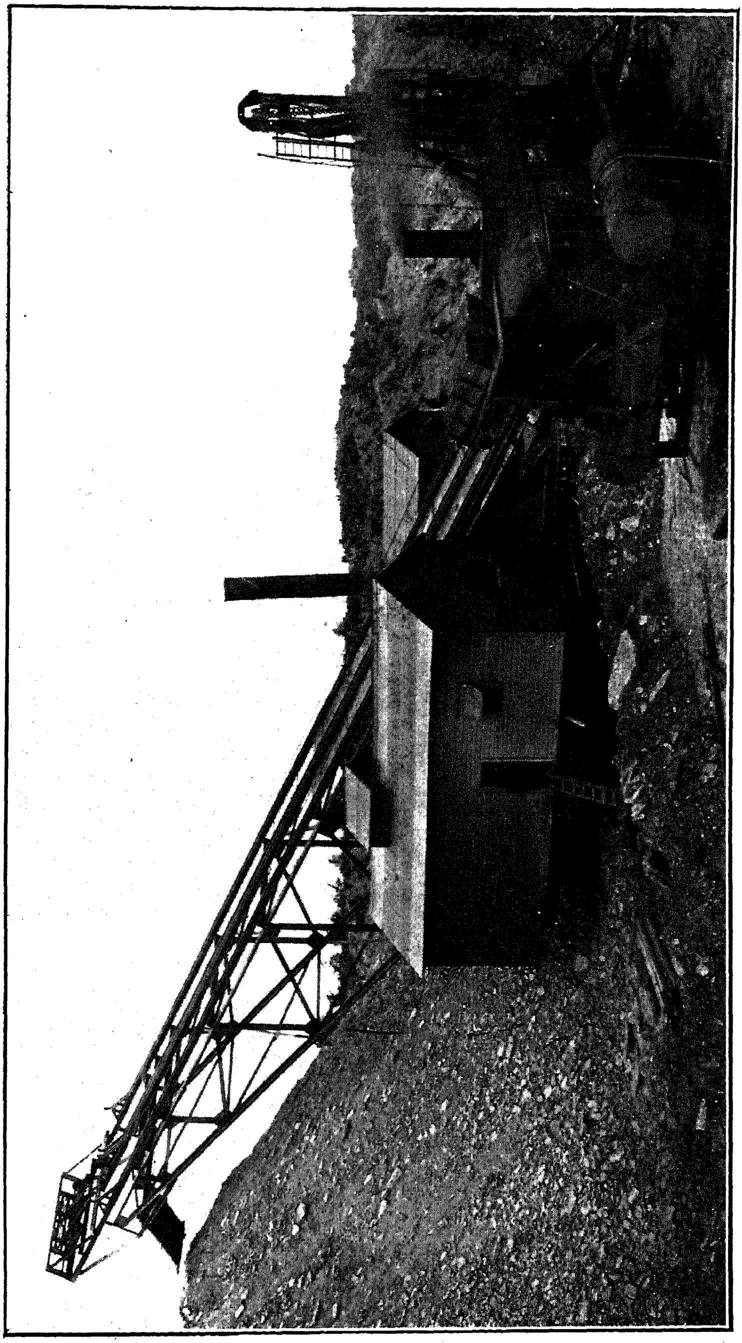


FIG. 1.—STEAM-SHOVEL AND PORTABLE HOIST USED IN EXCAVATION, ONTARIO, N. Y.

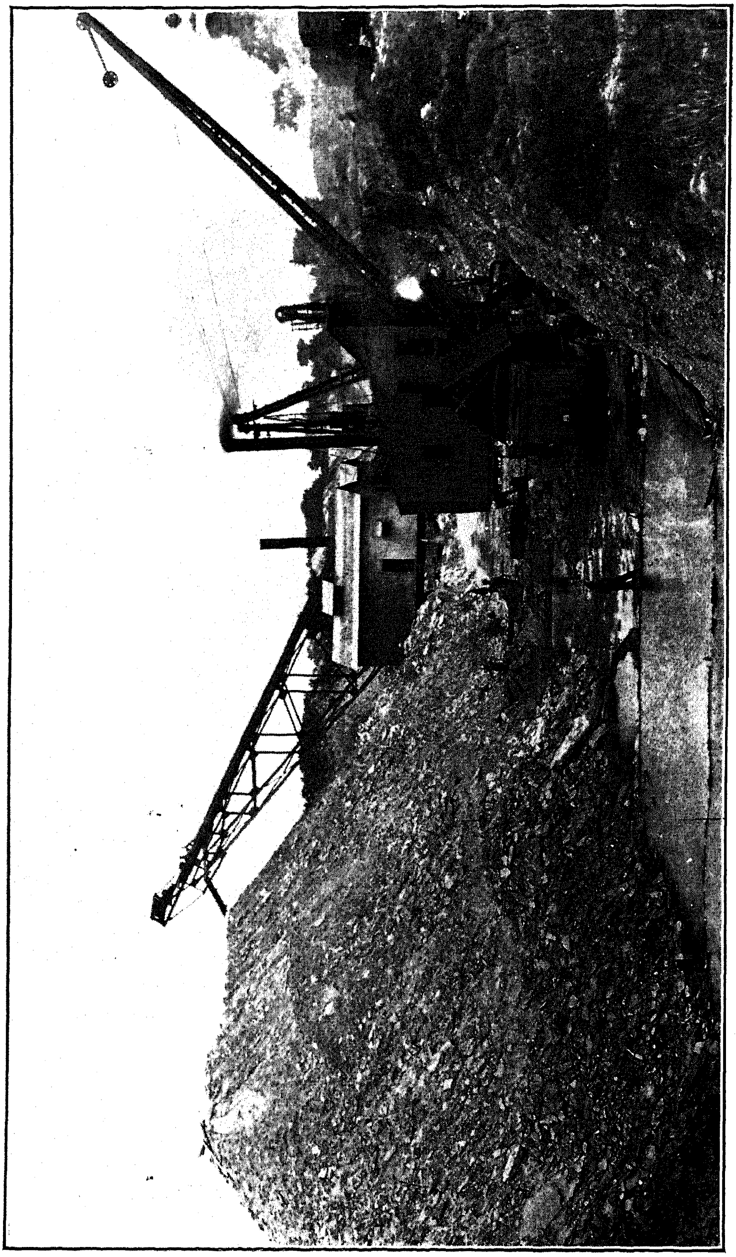


FIG. 2.—REMOVING THE ORE FROM TRENCH, ONTARIO, N. Y.

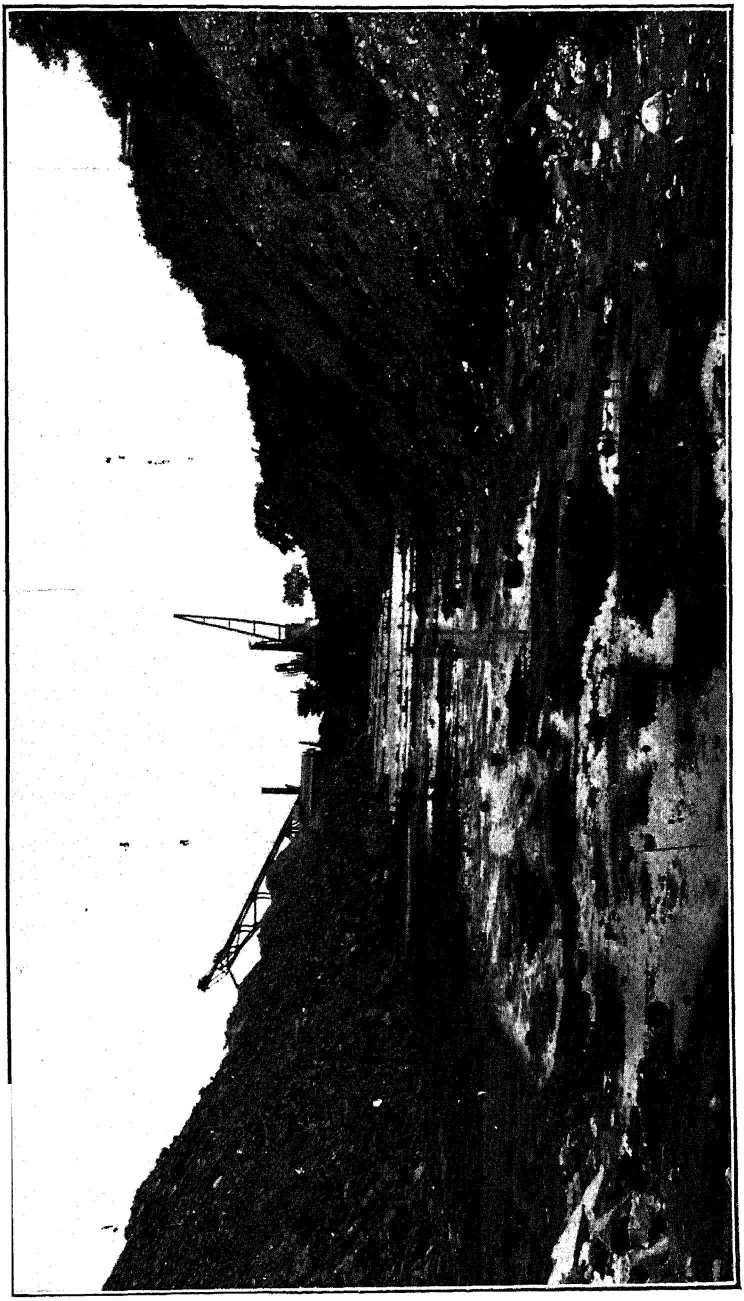


FIG. 3.—TRENCH AFTER REMOVAL OF ORE, ONTARIO, N. Y.

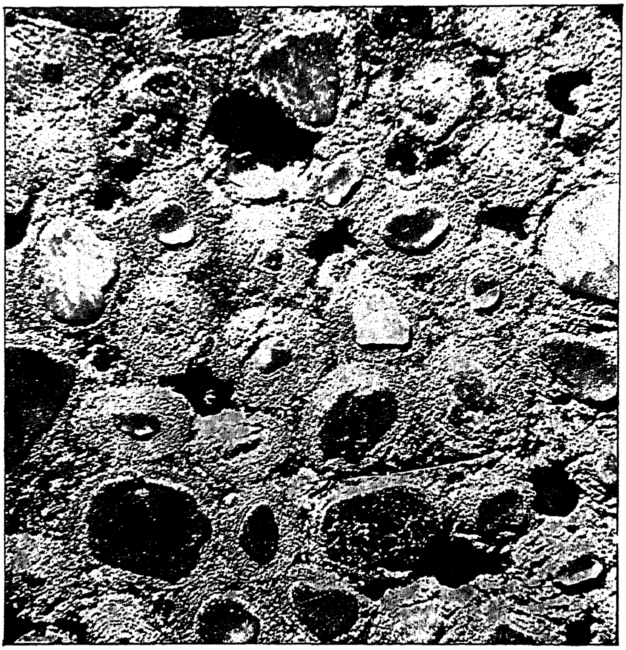


FIG. 4.—PHOTOMICROGRAPH OF OÖLITIC ORE, SHOWING CONCENTRIC STRUCTURE.

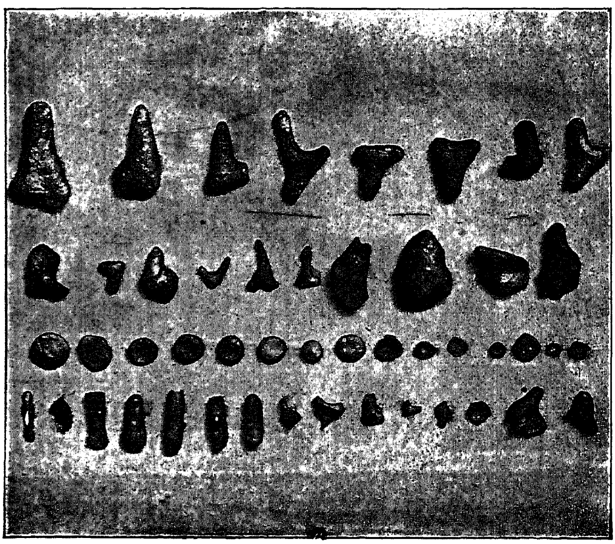


FIG. 5.—FRAGMENTS OF BRYOZOANS AND CRINOIDS FROM THE FOSSIL CLINTON ORE.

Underground flowage, either vertically or horizontally, is very limited in amount, owing to the attitude and character of the beds. As has been stated, the latter lie nearly horizontal, and at no time since their emergence from the sea have they been highly inclined. Moreover, the principal element in the formation is a fine, closely-compacted shale that is practically impermeable to water. If there had been any considerable movement of water through the beds, its influence should be discernible in the decomposition of the shale and in leaching and solution of its contained lime and the limestone-layers; but except on the immediate outcrop, where they are exposed to weathering, the strata show no such effects. In drilling-operations it is a frequent experience to encounter natural gas in the shales, even within 100 ft. from the surface, but very little or no water.

In Cayuga and Wayne counties the main ore-seam is always covered directly by limestone, and in places it is underlain also by limestone. It is hardly conceivable that ferruginous solutions could pass through the outer beds of limestone, leaving them unreplaced, to deposit their burden in the interior, forming a solid and sharply differentiated stratum of ore.

The oölitic hematite as found at Clinton and vicinity, on the other hand, occurs within shales and shaly sandstones. The hematite is deposited layer upon layer about a nucleal grain, usually of well-rounded quartz. Its formation has involved no chemical interchange, as in the case of the fossil ore, but the quartz has supplied simply a convenient surface for precipitation of the iron, similar to its function in the deposition of oölitic limestone. Professor Smyth[9] has called attention to a peculiar feature of the oölites in the presence of amorphous silica, apparently deposited at the same time and from the same solution as the iron. The silica-layers are not discernible to the eye in examining thin sections of the ore, but appear when the latter is subjected to the action of hydrochloric acid. After the hematite is removed in solution, there remain perfect casts of the original oölites in transparent gelatinous silica.

The two structures—fossil and oölitic—are fairly well contrasted in most cases, so that samples from any particular

---

[9] *Op. cit.*

locality usually have a uniform appearance, yet oölitic grains can be found almost always, distributed through the mass, even in ore that at first looks to be of purely fossil nature. The deposition of iron in layer after layer can scarcely be explained as taking place elsewhere than in bodies of standing water, with the nucleal grains free to roll about and in complete contact with the ferruginous solutions.

The general features of the hematite-beds, their uniformity of character, constancy of horizon, and persistence over wide areas, as revealed by field-examination, are thoroughly in agreement with the view of their sedimentary accumulation.

Fig. 4 is a photomicrograph of oölitic ore, showing the concentric structure, and Fig. 5 shows fragments of bryozoans and crinoids from the fossil Clinton ore.

Enrichment by solution and redeposition of the iron has not occurred in the New York beds. Whatever variation in iron-content there may be, and it is usually small, is to be considered original or due to weathering on the surface. There are no bodies of soft ores at all comparable with those found in the Southern districts. This may be ascribed in large measure, perhaps, to the effects of the glacial invasion, for, in the long period previously in which the beds were exposed to atmospheric agencies, it seems likely that the ores may have been weathered back some distance from the outcrop, but the weathered portions were planed off by the ice in its southward advance. The chief effect of weathering is the removal of calcite which cements the particles of hematite. This may cause a noticeable increase in the iron-percentages.

The iron was probably carried in solution as carbonate, and was precipitated in the form of ferric hydrate or limonite. The subsequent change to hematite was brought about under the influence of heat and pressure while the beds were buried beneath great thicknesses of overlying strata that were later eroded away.

It seems quite certain that the ores must have been deposited in shallow lagoons or basins temporarily shut off from free communication with the great interior Palæozoic sea. Under other circumstances, the iron brought down with the wash from the land could hardly be collected and concentrated in the waters sufficiently to make possible its deposition on so ex-

tensive a scale. The influence of bacteria may have been a factor in oxidizing and precipitating the iron, as remarked by Loomis,[10] who has described certain microscopic fungi from the ore at Rochester. From a study of the fossil forms in the ore, Loomis[11] also expresses the view that the different species are below the average normal size, and attributes this to a retardation of development by the ferruginous character of the waters in which they lived.

The source of the iron, as well as of the accompanying materials, is to be found in the pre-Cambrian land masses of the Adirondacks and the Canadian highlands. The crystalline rocks of these regions uniformly carry several per cent. of iron oxides, both free as magnetite, and combined in the silicate minerals, and in the Adirondack region they inclose important bodies of magnetite, hematite, and pyrite. The Clinton belt has its maximum development in the stretch from Clinton to the west end of Oneida lake, where there was apparently an embayment curving around the southwestern border of the Adirondacks. The present outcrop in this part is everywhere within 50 miles at most of the exposed crystalline rocks. Farther west the beds diminish gradually with the increase of distance from the Adirondacks, and in the extreme west the materials probably came from the Canadian pre-Cambrian area. East of Clinton there is a rapid thinning of the beds, since the old Appalachian highland that limited the Clinton sea in this direction is soon reached.

---

[10] *Bulletin No.* 39, *New York State Museum.* p. 223 (1900).

[11] *Fifty-Sixth Annual Report, New York State Museum,* vol. ii., p. 895 (1904).

Ozark Lead- and Zinc-Deposits: Their Genesis, Localization, and Migration.

BY CHARLES R. KEYES, DES MOINES, IOWA.

(Chattanooga Meeting, October, 1908.)

CONTENTS.

I. Introductory.

Industrially, the most important service that geological science can now render to mining in the Upper Mississippi lead- and zinc-fields is to devise some practical scheme whereby new productive ore-belts may be located with greater certainty than they have been in the past. Up to the present time there has been little real aid of this kind, and in spite of all that has been worked out concerning the geologic features of the region, and notwithstanding all that has been written about these ore-deposits, this particular phase of the subject has been, for some reason or other, the very one that has received the least consideration. The neglect by geologists in this respect has been all but complete. The following notes are suggestive of some practical solutions to the problems presented.

As here depicted, the geologic relationships of the ore-bodies were surmised some years ago, though somewhat vaguely perhaps. No opportunity for securing critical data appeared until recent investigations of a private character were undertaken, which had in view the extensive prospecting of undeveloped areas. Fresh from rather wide experiences of a decade in the principal mining-regions of four continents, and after a lenstrum's residence among the chief mining-camps of the West and of Mexico, under conditions which measured geologic hypotheses in meters and rigidly tested their values in terms of dollars, the recent return to the Joplin field and a new survey of its ore-conditions were fraught with unexpectedly suggestive results. Aspects of the ore-deposition were presented of which previously there was no hint.

Ten years' absence from the district had tended greatly to clarify former hazy impressions. It enabled many features, then only dimly outlined, to be considered more nearly in their true perspective and in their real relationships to their surroundings. It permitted a great mass of details collected a dozen years before in connection with the work of the Missouri Geological Survey to be viewed under new light and to

be again rapidly brought together into a connected whole. Many problems that formerly seemed formidable no longer appeared puzzling.

In the past dozen years there have been numerous contributions to the already voluminous literature on the Ozark ore-deposits. It is to certain points in these discussions that attention is now also directed.

The lead- and zinc-ores of the Ozark region have recently been brought into scientific prominence by reason of the part which they are made to play in the far-reaching generalizations which Van Hise and Bain have formulated for the genesis of ore-deposits in general. This region thus becomes the type-locality and the best-known illustration selected to illuminate the fundamental principles of their purely deductive hypothesis of a first concentration of ore-materials by upward circulations and a commercially more-important second concentration by descending percolation of mineral-laden waters. Upon the facts presented by this region it seems that their entire theory must stand or fall.

Missouri lead- and zinc-deposits have been for many years also the subject of extended controversy between the "ascensionists" and the "lateral-secretionists" of the United States. The literature which has, in consequence, developed forms a distinct branch of our general knowledge of ore-deposits. Almost all of these memoirs and articles relating to the district contain much that is of permanent value; some of them present much that is especially suggestive and helpful; a few of them include the germs of generalizations which are applicable not to Missouri alone, but to ore-deposits throughout the world.

In all of the discussions and descriptions there appear to be a number of important relationships of the ore-bodies that have been overlooked. On the other hand, isolated facts are recorded which have much deeper significance than any direct suggestions yet made. And there are some assumptions which should no longer find tolerance.

II. Hypotheses of the Origin of Ozark Ores.

1. *General Considerations.*—While there are many accounts of the different mining-camps, and descriptions of local features

concerning the ore-deposits, the broader considerations are reducible to half a dozen distinct hypotheses of genesis. In order that they may be more clearly understood in their relationships to the discussions which follow, the essential features of these six hypotheses may be briefly noted here, and the points in which they differ from one another emphasized.

2. *Schmidt-Leonhard and Gage View.*—In describing the southwestern Missouri zinc-district, Schmidt and Leonhard,[1] directly opposing Percival's well-known views, regard the lead- and zinc-ores as deposited from solution at the same time that the limestones were undergoing dolomitization. On account of the latter change there were:

"Disturbances and ruptures in the Chert in consequence of the contraction of the Limestone during the metamorphic action. . . . This metamorphic action was confined to a part of the alternate layers of Limestone and Chert, and very limited in its vertical extent—rarely exceeding 20 feet."

Ore-deposition, however, continued long after dolomitization ceased.

In his notes on the southeast region, Gage[2] distinguishes four classes of lead-deposits: disseminated beds, flat sheets, vertical fissures, and horizontal fissures, the principal division of which is the "cave formations." After briefly discussing the origin of the caves of the region through solution and removal of the softer and more porous layers of limestone, he says:

"After the formation of these caves, which were probably at first open, water passed through, slowly refilling them by depositing the various minerals held in solution. It is possible that the metalliferous deposits were formed synchronously with the formation of the caves, as the same water which dissolved the rock away may have also, at the same time, held the various minerals in solution." (P. 615.)

3. *Jenney Opinion.*[3]—This is the first direct application to the Joplin district of the idea, unqualified, that the ores were derived immediately from the heated centrosphere and reached the surface by means of profound faults.

"The evidence obtained in this investigation indicates that the ores and the associated minerals have all been deposited from aqueous solutions, probably of moderate or normal temperature and pressure, and that the fissures connected with

[1] *Missouri Geological Survey Report* 1873–4, p. 412 (1874).

[2] *Missouri Geological Survey Report* 1873–4, p. 612 (1874).

[3] *Trans.*, xxii., 171 to 225 (1893).

the ore-bodies have formed the channels through which the mineralizing waters were introduced. It is also evident that the lead and zinc were not derived from the geological formations in which the deposits occur, or from the overlying or underlying sedimentary strata, but that the source of the metals was exotic and was probably deep-seated in the primitive rocks." (P. 215.)

"The only theory which, in all observed instances, will account for the occurrence of the deposits of lead- and zinc-ores and the associated minerals in the Upper and Lower Mississippi region is that of ascension, the source of the metals existing deep in the primitive rocks. With the discovery that the ore-bearing crevices are faulting-planes of indefinite vertical extension, the classification of the deposits of the Mississippi valley as the fillings of 'gash veins,' or crevices formed by the contraction or shrinkage of rocks, and confined to a narrow vertical range within the geological horizon, must be abandoned." (P. 223.)

The same author assumes a general law that

"all workable deposits of ore occur in direct association with faulting fissures traversing the strata, and with zones or beds of crushed and brecciated rock, produced by movements of disturbance." (P. 184.)

From a general view-point, Jenney's theory of the ore-deposition is a natural and very satisfactory explanation.

4. *Pošepný Idea.*—In his general essay on ore-genesis, Pošepný[4] makes extended references to the southeastern Missouri lead-deposits which he visited. He regards these ore-deposits as primary xenogenous fillings; that is, original pores and cavities in the dolomites filled from deep-seated solutions.

"The deposits occurring near the 'islands' of granite and porphyry, have special interest. While the Silurian limestones of the surrounding country, farther from these islands, present chiefly only lead- and zinc-ores, other metals, such as copper, cobalt, and nickel, occur as the Archæan foundation-rocks are approached; and this circumstance is, to my mind, an indication that the source of the lead-deposits also is to be sought in depth."

From a general view-point, Pošepný's theory of the ore-deposition is also a natural and very satisfactory explanation.

5. *Winslow-Robertson Hypothesis.*[5]—In its briefest terms, the idea advocated is,

"Original diffusion through the country rocks and subsequent concentration through surface decomposition of the latter, supplemented by percolating waters."

In support of the hypothesis, the long tongue of Early Carboniferous limestone extending eastward from Joplin into the

[4] *Trans.*, xxiii., 301 (1893).
[5] *Missouri Geological Survey*, vol. vii., p. 469 (1894).

center of the Ozark dome is regarded as an ancient estuary of the sea, into which metallic solutions from the surrounding land were concentrated more abundantly than elsewhere on the neighboring sea-coast. The later extensive surface-erosion is considered as liberating the generally-diffused metallic particles, which were carried directly below and precipitated in the crevices and open spaces. From a general view-point, Winslow and Robertson's theory of the ore-deposition is a natural and very satisfactory explanation.

6. *Branner Proposition.*—In its essential features, it is stated [6] that:

> "The zinc was originally deposited in sedimentary beds, mostly of organic origin, in which much of it is still found.
>
> "The growth of the crystals in the original bedded-deposits took place prior to the hardening of the enclosing silts."
>
> Subsequent changes have taken place whereby "vertical and other fissures have been filled with ores brought into them by circulating waters from above, from below, and from the sides."

The first-mentioned class of deposits is considered as contemporaneous with the formation of the rocks themselves; the second class as secondary deposits, following geologic structures. From a general view-point, Branner's theory of the ore-deposition is a natural and very satisfactory explanation.

7. *Keyes Statement.*—In the reports of the Missouri Geological Survey and the various later articles growing out of the work of that organization, it is shown that in so far as the lead- and zinc-deposits of the Ozark region are concerned there are no less than four distinct modes of ore-genesis and a number of distinct and distant periods of ore-formation. Regarding the Joplin, or southwestern Missouri, area,[7] it is said that extensive observation had failed to disclose in the region any undoubted evidences of marked faulting. The physical conditions are peculiar in that the Ozark region appears to be rising much more rapidly than the surface-waters can degrade the country. As a result, much of the water that falls upon the surface is carried off in deep, underground channels down the slope of the great dome.

> "The subterranean watercourses are subject to the same laws of aggradation as the rivers open to sky. Barriers arise which cause the cavernous courses to silt

---

[6] *Arkansas Geological Survey, Annual Report* 1892, vol. v., p. 34 (1900).

[7] *Trans.*, xxxi., 608 (1901).

up, as it were, often to the extent of hundreds of feet. The local groundwater-level for the time rises accordingly. Within a range of several hundreds of feet there may be thus no discernible relationship between the groundwater-level and the character of the ore."

Without inquiring into the original source of the ores, it is considered[8] that

"ore concentration does not generally take place directly through surface decomposition of rock-masses, in areas such as the Ozark lead and zinc region. . . . if it be assumed that all rocks at all times contain ample supplies of metallic salts in a diffused condition sufficient for the most extensive ore deposits, and that the circulatory waters hold them at all times in solution to a greater or less but adequate extent, these must be regarded as general, always present conditions. But concentration of metallic salts into ore bodies must be admitted to be accomplished under special local conditions of geologic structure, quite independent of all local rock decomposition and land degradation."

The theory is one of mainly vadose circulation and ore-concentration in impounded, underground waters under definite conditions of geologic structure.

8. *Van Hise-Bain Theory.*—On the close agreement of theory and fact concerning ore-deposition in the Joplin district must rest the foundations of the hypothesis of a distinct and notable first concentration by ascending waters and a second concentration through descending circulation.[9] Accordingly,

"the waters were in circulation, the motive power being gravitative stress, and the deposition occurred mainly as a result of the mingling of solutions, the sulphate solutions of the Cambro-Silurian passing up through the Eureka-Kinderhook, along fault-planes, and mixing with the reducing solutions of the Carboniferous Limestones. The ores were originally deposited as sulphides, and have since been reconcentrated by the downward flow of surface-waters with the production of an upper surface-zone of carbonates, silicates, etc., and a middle zone of enriched sulphides. Another result has been the production of an orderly vertical arrangement of the sulphides."

From a general view-point, Van Hise and Bain's theory of the ore-deposition is a very satisfactory explanation.

III. Requirements of an Adequate Working-Hypothesis.

In accounting for the deposition and localization of any ore-bodies it is not a hypothesis but the hypothesis that is sought.

---

[8] *American Geologist*, vol. xxvii., No. 6, p. 362 (June, 1901).

[9] *Transactions of the Institution of Mining Engineers* (London), vol. xxiii., p. 401 (1901-2); also, 22*d Annual Report, U. S. Geological Survey*, pt. ii., p. 23 (1901).

A possible explanation of local ore-genesis is usually of small practical value. From a general view-point, such a theory of ore-deposition may be a natural and very satisfactory general explanation, and still not be at all applicable to the locality under immediate consideration. It is not sufficient to devise a plausible explanation of natural phenomena, for this can satisfy merely idle curiosity.

Generalization is useful only when it does not ignore essential facts. Any theory, in order to be generally acceptable, must be able to stand the most rigid tests as to its local applicability. One of the crucial tests is measured by its value in prognostication. It is necessary, if it is to command respect, that a hypothesis accurately predict results. In proportion as it does this is a hypothesis of service in mining-operations. In the Ozark region these considerations have not received the critical attention that they deserve, or that practical experience in mining-exploration demands.

An adequate hypothesis of ore-formation in a region such as the Ozarks should, among many other features, clearly and satisfactorily explain:

1. Genetic class of the ore-bodies.

2. Whether ore-segregations are shallow or deep formations.

3. Definite relationships of the ore-bodies as a whole to the major geologic features of the region.

4. Direct relations of the ore-bodies to the local geologic structures, in sufficient detail for the conclusions deduced to be of use in prospecting for unknown deposits.

5. Dependence of ore-concentration and localization upon the direction of general ground-water circulation.

6. Depth, both possible and probable, of workable and available ore-bodies.

7. Control of ore-bodies by fault-planes, jointing, or other lithoclasic features.

8. Geologic age, within narrow limits, of the present ore-accumulations.

9. Indication of an adequate immediate source of the ore-materials, as a check on the probable results of underground circulations.

10. Conditions and extent of the secondary enrichment phenomena.

11. Peculiarities and possible oscillations of the local ground-water level.

12. Plausible phases of other hypotheses advanced.

In the present connection these various features need neither be equally discussed nor the themes symmetrically developed.

IV. General Geotectonics of the Ozark Region.

1. *Salient Aspects.*—In their direct genetic bearing upon ore-deposition the larger features of geologic structure have a significance that does not appear to have been made the most of during past investigations of the region.

In its simpler aspects, the Ozark uplift may be considered as a low quaquaversal dome 500 miles long by 200 miles broad. Centrally it rises about 1,400 ft. above the level of the plains-country around the margins.[10] It is a region that has been repeatedly upraised and planed off, until in the middle portion the oldest known rocks only are exposed. The last planation, apparently down nearly to sea-level, seems to have occurred in Late Tertiary time. The present dome is manifestly the result of very recent and very rapid upward movement. This is a feature that has been wholly overlooked in all of the discussions of the ore-deposits. It appears to be one of the most important factors influencing ore-concentration and ore-deposition.

A diagrammatic sketch of the geologic structure of the Ozark dome is shown in Fig. 1.

2. *Geologic Formations.*—The rock-terranes to be taken into account in considering the ore-deposits of the Ozarks are merely (*a*) the Cambrian dolomites and the Ordovician sandstones and dolomites, (*b*) the Devonian shales, (*c*) the Early Carboniferous limestone, and (*d*) the Coal Measures. Their characteristics and local peculiarities have been frequently described in the various reports of the Missouri Geological Survey and other publications. No special consideration of these formations is, therefore, necessary at this time.

3. *Character of Faulting.*—Of the structural features presented by the Ozark region, faults, minor flexures, gentle warpings, and unconformity-planes have had an influence upon the localization of the ore-deposits that is unusual in many ways.

---

[10] *Missouri Geological Survey*, vol. viii., p. 321 (1895).

In the main, the faults are unimportant, infrequent, and of small throw; while ore-bodies are not very often genetically connected with them. This fact is, indeed, very remarkable, especially in view of the usual rôle which these structures play. Folding, warping, and unconformity phenomena have had upon ore-deposition an influence the importance of which is more and more appreciated as the field is more thoroughly and critically investigated.

In the various geological reports on Missouri,[11] attention is repeatedly called to the rather remarkable character of the faults in the ore-producing districts. In the southwestern district there were obtained no conclusive evidences of dislocations of more than a few feet. The extensive rock-shattering associated with most of the ore-bodies was found not to be fault-breccias at all, but breccias due entirely to other causes. In other parts of the Missouri region where extensive faults are known to occur there is no ore associated. This is notably the case with the great fault in the vicinity of Mine La Motte. The displacement here is over 200 ft., but there is no ore whatever in the immediate neighborhood.[12] The Cap-au-Grès fault, at the north base of the Ozark dome, near the mouth of the Missouri river, has a throw of more than 1,000 ft., yet there are no ore-deposits of workable character near it.[13] The extensive faulting in the Joplin district which has been lately described has been found by still more recent critical inquiry to be chiefly unconformity phenomena, thus fully substantiating the early observations. So far, then, as any dependence of ore-bodies upon fault-lines is concerned, the association of the two may be altogether neglected.

4. *Extent of Flexing.*—The existence of well-defined, but gentle, flexures in the strata, the axes of which pitch radially down the Ozark dome, and their definite association with producing ore-belts, have been known for years. However, they are features which have not yet received consideration in any of the discussions of the Missouri districts. One in particular of these gentle folds which passes through Sedalia has long been recognized. Others have been noted in the Osage River

[11] *Missouri Geological Survey*, vols. i.–xii. (1891–1897).

[12] *Missouri Geological Survey*, vol. ix., pt. iv., p. 62 (1896).

[13] *Proceedings of the Iowa Academy of Sciences*, vol. v., p. 58 (1898).

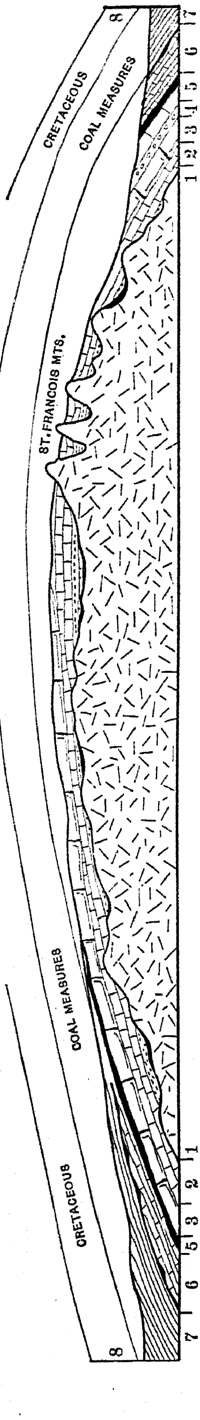


FIG. 1.—GEOLOGIC STRUCTURE OF THE OZARK UPLIFT IN MISSOURI.

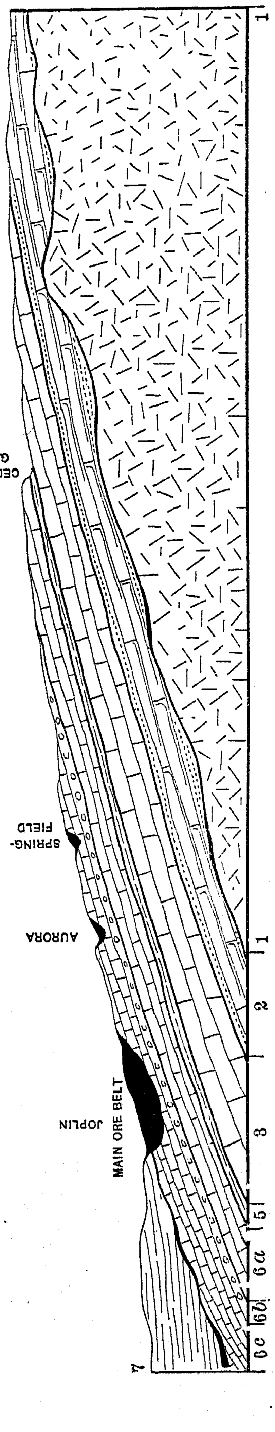


FIG. 2.—POSITION OF ORE-BELT AT BASE OF OZARK DOME.

section by Keyes,[14] and by Winslow and Robertson.[15] More recently, Branner[16] has noted several small flexures in the northern Arkansas mineral district that seem to have close relationship with the ore-deposits.

In the central Missouri district it has been lately discovered that all of the known mines are located in certain straight and narrow belts, which are near and parallel to the axes of shallow synclines. These synclines pitch radially from the center of the Ozark uplift. The Joplin, or southwestern, district appears to lie wholly within the limits of a broad syncline trending directly westward. Its position is well shown on the geologic maps of the region in the long tongue of Early Carboniferous limestones which extends eastward from the Kansas border nearly to the present crest of the Ozark dome. The preservation of numerous outliers of the Coal Measures high upon the back of the dome seems also to be due to their protection from general erosion on account of local synclinal structure of the strata.[17]

5. *Unconformity Irregularities.*—Of like significance in their bearing upon ore-deposition are the troughs and basins which lie in planes of unconformity. The basal plane of the Coal Measures is of this character. But most of the deep, débris-filled gorges are now believed to be solution-channels of which the roofs have fallen in. In the southeast Missouri district the unconformity-plane marking the base of the Cambrian terranes seems to have largely controlled the localization of the great bodies of disseminated ores.[18] Many of the borders of the gorges at the base of the Coal Measures have been mistaken for fault-lines; but it appears that nowhere in the Ozark region has faulting had very much to do with the localization of ores. On the other hand, most, if not all, of the ore-deposits occur in slightly pitching basins or troughs.

6. *Warping of Strata.*—As ordinarily considered, the syncline and anticline are represented only in two dimensions. If viewed as a geologic structure of three dimensions, there is obtained an entirely different conception of the phenomenon. The effect

---

[14] *Bulletin of the Geological Society of America*, vol. xiii., p. 267 (1901).

[15] *Missouri Geological Survey*, vol. vi., p. 359 (1894).

[16] *Arkansas Geological Survey*, *Annual Report* 1892, vol. v., p. 32 (1900).

[17] *Missouri Geological Survey*, vol. x., p. 341 (1895).

[18] *Missouri Geological Survey*, vol. ix., pt. iv., p. 58 (1896).

may be compared to the waves of the sea. Warping in this sense is not commonly described in treating of geologic structures. Nevertheless, the phenomenon is widely observable. When of minor character it is difficult to measure, usually for want of precise topographic maps and lack of sufficient detailed data.

Minor warping has long been recognized in Missouri and its local influence upon ore-deposition surmised. The best illustration of the effects of minor, fine-patterned warping of strata is in the Joplin district. It is graphically shown in the underground-contour map of the surface of the Grand Falls chert-bed recently published by Smith and Siebenthal.[19] The remarkable relationships which the ore-bodies bear to the depressed areas are fully described later on.

V. Classes of Ore-Deposits.

As recently shown,[20] there occur in the Ozark region no less than four distinct types of lead- and zinc-deposits. They are: (*a*) fumarole impregnations, as in the case of Silver Mines, east of Ironton, as described in considerable detail by Haworth;[21] (*b*) true fissure-veins, as in the Ouachita region, of which Comstock[22] has given us such a full account; (*c*) direct precipitative action in connection with organic matter, as exemplified by the sporadic occurrences of zinc-blende in the Carboniferous sandstones near St. Louis, noted by Wheeler,[23] and in the coal-beds of Morgan county, Mo.;[24] and (*d*) impounded deposits of the vadose zone, to which nearly all of the workable ore-deposits of the region belong.

The first three types may be passed over altogether in the present connection, and the attention directed wholly to the problems bearing upon the fourth class. As will be seen hereafter, the recognition of ores of the first circulation and of a subsequent second concentration need not enter into consideration. All the facts seem to point to the conclusion that the effects of a first concentration from deep-seated waters are a

---

[19] *Geologic Atlas of the United States*, Folio No. 148 (1907).
[20] *Trans.*, xxxi., 606 (1901).
[21] *Missouri Geological Survey*, vol. viii., p. 81 (1895).
[22] *Arkansas Geological Survey*, *Annual Report* 1888, vol. i., p. 219 (1888).
[23] *Transactions of the St. Louis Academy of Science*, vol. vii., p. 123 (1895).
[24] *Trans.*, xxx., 346 (1900).

negligible quantity, and, from a practical view-point, unimportant.

As I understand Van Hise and Bain's idea of a first concentration in rocks to a small fraction of 1 per cent., the term general diffusion would be more fitting. It is not a special concentrating process or tendency at all. It is a universal condition of all rocks to a greater or less extent. Appreciable concentration can begin to take place only in the vadose zone near the level of ground-water. In any case, there must be certainly notable local accumulations of ore-materials before actual concentration can be considered. Outside of ore-bodies of deep-seated igneous character, which the majority of ore-deposits seem to be, I cannot, at present, see how a first concentration in the sense mentioned can occur.

VI. Vadose Ore-Concentrations.

For reasons elsewhere stated at length, the Ozark ore-deposits, as they exist to-day, are treated strictly as vadose concentrations, accomplished under definite tectonic circumstances. Accordingly, it is not thought that the ascension theory proposed by Percival,[25] Jenney,[26] Pošepný,[27] and later, in compromised form, by Van Hise and Bain,[28] finds support in any recorded observations.

In its broader conception of deriving the ore-materials from the surrounding rock-masses, not necessarily near the ore-bodies themselves, but laterally rather than from strata below, the lateral-secretion theory of ore-accumulation, as advanced with various minor differences by Whitney,[29] Chamberlin,[30] Winslow and Robertson,[31] and Branner,[32] appears to find in all of the lately-acquired evidence of critical character the fullest confirmation.

As I take it, Sandberger's idea[33] of the lateral-secretion

[25] *Wisconsin Geological Survey*, 1st *Annual Report*, p. 30 (1855).

[26] *Trans.*, xxii., 219 (1893). [27] *Trans.*, xxiii., 301 (1893).

[28] *Transactions of the Institution of Mining Engineers* (London), vol. xxiii., p. 401 (1901–2).

[29] *Wisconsin Geological Survey*, vol. i., p. 398 (1862).

[30] *Geology of Wisconsin*, vol. iv., p. 544 (1882).

[31] *Missouri Geological Survey*, vol. vii., p. 467 (1894).

[32] *Arkansas Geological Survey*, *Annual Report*, 1892, vol. v., p. 34 (1900).

[33] *Untersuchungen über Erzgange*, p. 17, *et seq.* (Wiesbaden, 1882).

theory was that transportation of the ore-materials took place mainly in what we now call the vadose zone. This author doubtless made some mistakes in not properly referring some ore-deposits, as in the Freiberg and Pribram districts, for example, and some of his methods of obtaining concrete proofs of his contentions were probably not so convincing as they might be, but he probably did not really hold such extreme views as some of his critics would have us believe. The conception was already a century old when Sandberger took it up, as is shown in the quaint and curious writings of such authors as Delius,[34] Gerhard,[35] Lasius,[36] and others of their time; it also had the support of such authorities as the great Bischof,[37] Wallace,[38] and Forschammer[39] a quarter of a century before Sandberger's work appeared. The great influence of lateral secretion in the localization of enrichments has been lately emphasized by Weed[40] especially.

Regarding the Ozark ores, the thesis here presented is that the principal deposits are the ultimate results of vadose migrations of ore-materials, down structural troughs pitching with the slope of the Ozark dome, and localized in minor basins through impoundment against the continually-retreating margins of the Coal Measures. This view of the ore-deposits differs from the various others chiefly in that it regards localization as dependent directly and entirely upon conditions imposed by well-defined peculiarities of geologic structure.

If they could be properly compared with true fissure-veins, the Ozark deposits would correspond practically only to the narrow zone of secondary enrichment. This enriched zone, however, instead of following directly downward the course of nearly vertical veins, is continually migrating almost laterally down the stratigraphic slope of the dome. As zones of enrichment, the principal Ozark ore-bodies have relatively little oxidized ores above and practically no original sulphide ores below.

---

34 *Von dem Ursprung der Gebirge u. d. darinnen befinden Erzadern* (Leipzig, 1770).

35 *Beiträge z. Chemie u. Geschichte des Mineralreiches* (Berlin, 1776).

36 *Beobachtungen über die Hartzgebirge* (Hanover, 1790).

37 *Lehrbuch der chemischen und physikalischen Geologie*, 2d ed., vol. iii., p. 665 (1866).

38 *The Laws which Regulate the Deposition of Lead Ore*, etc. (London, 1861).

39 *Poggendorff's Annalen der Physik und Chemie*, vol. xcv., No. 5, p. 60 (1855).

40 *Bulletin of the Geological Society of America*, vol. xi., p. 179 (1899).

In this respect there is direct antithesis to the hypothesis urged by Van Hise and Bain.

VII. Geographic Distribution of the Ore-Bodies.

It is probably chiefly on account of the manner, more than anything else, of the descriptive treatment of the ore-bodies on the basis of the productive camps that the general idea has gone abroad that their distribution is very limited, very local, and confined to widely-separated areas. This notion has generally prevailed not only among those not personally acquainted with the region, but among those who have been long familiar with the various districts. In many ways the conception is very misleading. Industrially, the effect of the prevalence of such a notion is particularly unfortunate. Perhaps no single factor has been so largely instrumental in retarding the general and natural expansion of mining-activities throughout the region.

In reality, the ore-deposits of the Ozark region are not limited to a few circumscribed areas. They extend far beyond the boundaries of the present recognized districts. Doubtless the ore-fields outside of the limited districts will prove to be much more extensive than the fields developed up to the present time.

Areally, the lead- and zinc-deposits of the Missouri-Arkansas region are mainly distributed in a belt of greater or less width which borders the basal margin of the Ozark dome and completely encircles it. This mineral-bearing belt, at its outer margin, is rather sharply delimited by the attenuated edge of the Coal Measures. Its inner margin is somewhat less definitely drawn, owing to local geologic features, to which reference is made farther on. There are, of course, some unimportant deposits within the main ring in positions suggesting former locations of the mineral-belt before the Coal Measures margin had moved so far as at present down the slope of the dome.

The character and position of the Ozark ore-belt, or ore-ring, is one of the most significant features bearing upon the genesis of the ore-deposits of the region. A cross-section of the dome, Fig. 2, shows the position of the main ore-producing belt with reference to the margin of the Coal Measures. Were the Ozark dome a segment of a perfect sphere, the distribution of the ore-materials at its base would be, doubtless, quite uniform. The

effects of crustal deformation have been unequal. In consequence of this, the accumulation of the ore-materials is also more or less unevenly disposed. At some points in the basal ring there are localized extensive deposits of workable character. These are separated by so-called barren tracts, or rather sections, in which ore-bodies are not, as yet at least, very important, or in which mining can be carried on only in a small way. Politically, this arrangement finds partial expression in the situations of the principal mining-camps.

In the usual descriptions of the lead- and zinc-mines it is customary to regard them as forming more or less distinct districts. In Missouri, for instance, there are three principal groups of mining-camps. The general recognition of these three mining-districts according to political boundary-lines rather than along geologic lines has entirely obscured whatever genetic relationships might possibly exist between the various groups of ore-deposits of contiguous camps. The title of southwestern district, for instance, has no further real significance than that of direction, being a vaguely-defined area lying somewhere in the southwestern part of the State of Missouri.

In a rough way, mining-men come to recognize in the several districts tracts of productive ground separated from one another by more or less broad strips of poor country, as it is called. Many miners associate the ore-deposits with the present stream-valleys. Thus far, also, all attempts of scientists to associate the ore-bodies with lines of faulting have been uniformly unsuccessful. There are a few exceptions. Nevertheless, there are discernible in the local distribution of mine-developments certain definite lines along which ore-bodies are more abundant and closer together than along other lines. Miner, layman, and scientist alike who have worked in the region are in accord with the idea that there must be some dependence of ore-occurrence upon geologic structure, even if it takes some of the refinements of investigation which the latter employs to determine what those relationships are. The local distribution of the ore-bodies along certain ascertained lines is more fully considered under the topic of the relationships to the geologic structures.

VIII. Relation of Ore-Deposit to Geologic Structure.

1. *General Occurrence of Ore-Bodies in Basins.*—Excepting the few isolated instances noted by Branner in Arkansas, the only plausible attempts to connect the Ozark lead- and zinc-deposits with definite geologic structures are those in which alleged faulting is regarded as the chief localizing-factor.

As already stated, marked faulting is not characteristic of the ore-bearing districts of the Ozark region. In fact, it is noteworthy on account of its general absence. Most of the profound faults described in connection with the ore-bodies have proved upon more critical examination to be stratigraphic disparities produced by old underground water-channels, often along the horizon of the unconformity-plane at the base of the Coal Measures. The joint-systems, when they will have been eventually made out, will be found no doubt to be also largely instrumental in initially determining the trunk courses of underground-water currents and channels.

So long ago as 1895, while I was in charge of the Missouri Geological Survey, shallow synclines and underground erosion-channels were recognized in the various mining-camps. With them were soon associated the principal ore-belts. At the time no opportunity was afforded to publish the details of the discovery in the reports of the Survey, although numerous incidental references were made to them. From a knowledge of this relationship several of the most important ore-bodies thus located were afterwards opened up. To this attention is called in another place. Were detailed cross-sections on selected lines available the principles involved would have, without question, a much wider practical bearing.

From a utilitarian stand-point, the recognition of the disposition of the Ozark ore-deposits in definite, basin-like areas has a far-reaching significance. It matters little as to the geologic origin of these ore-accumulating depressions. They may be true synclines, or structural troughs, as predominate in the central Missouri district, shown in Fig. 3. They may be occasioned by local faulting, as in some parts of the northern Arkansas district, as illustrated in Fig. 4. They may be inequalities in ancient erosion-surfaces, as in the southeastern Missouri lead-district, represented in Fig. 5. They may be due to both flexing and minor warping, as in the southwestern

Missouri district, according to Fig. 6. Or, they may be formed by the caving-in of underground channels, or caverns, and a

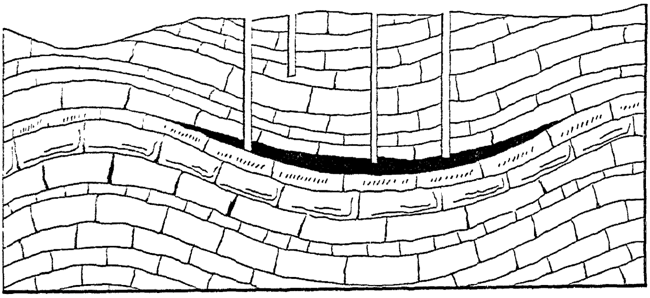


Fig. 3.—Ore-Bodies in Pitching Syncline on Osage River, in Missouri.

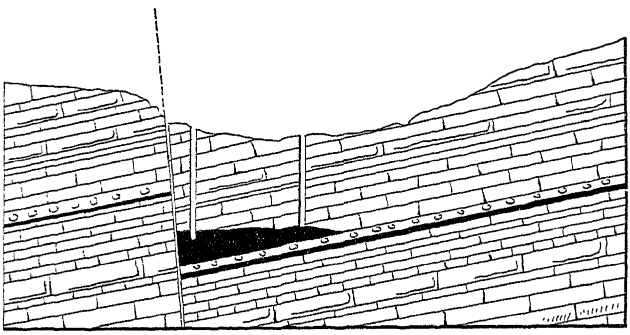


Fig. 4.—Ore-Bodies on Fault-Line in Sugar Orchard Valley, Arkansas.

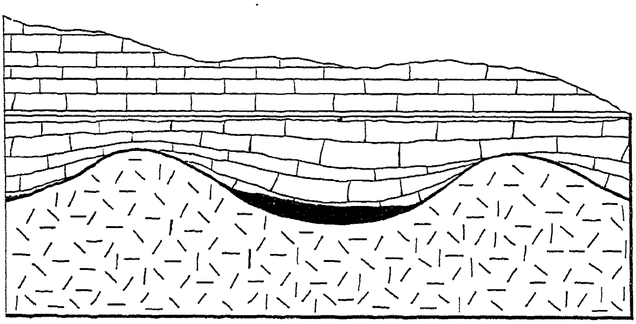


Fig. 5.—Ore-Bodies in Unconformity-Trough, Doe Run, Mo.

consequent damming of the outlet, as in many cases in the last-mentioned district, represented in Fig. 7.

In all of these examples it is especially noteworthy that the ore-deposits are definitely associated with, or localized by, geologic structures of some kind that produce in effect basins in which the underground waters are impounded, or their flow retarded.

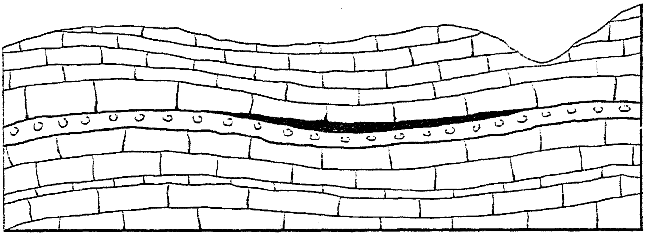


FIG. 6.—ORE-BODIES IN WARP-SAGS, NEAR JOPLIN, MO.

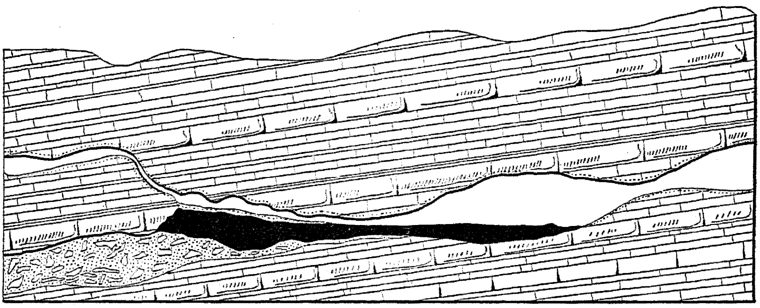


FIG. 7.—ORE-BODIES IN SILTED-UP CAVERN, NEAR AURORA, MO.

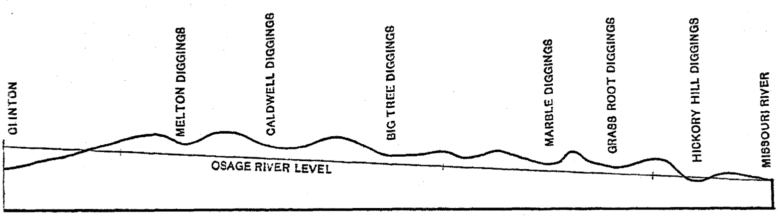


FIG. 8.—FLEXURES ALONG THE OSAGE RIVER, IN MISSOURI.

2. *Deposits of the Central Missouri District.*—The Osage river geologic cross-section, shown in Fig. 8, presents a very detailed picture of the geologic structure in a direction parallel to the margin of the Ozark dome and transverse to the distinct minor flexing.[41]

[41] *Missouri Geological Survey*, vol. vi., p. 359 (1894).

Although roughly segregated into camps, the scattered mines of the district appear, at first glance, to have little connection with one another, or not to have any plan of location or arrangement. Examined carefully with regard to the rather pronounced folding displayed clearly in the geologic cross-section mentioned, they are seen to arrange themselves along definite lines. These lines are all axes of synclines. The intervening barren tracts are found to be the anticlinal parts of the flexing. The most pronounced of these structural troughs extend from Rocheport to Vienna, from Tipton to Dixon, from Smithton to Richland, from Welton to Lebanon, from Warsaw to Phillipsburg, and from Clinton to Buffalo. Their trend is nearly SE–NW., and their pitch very closely corresponds to the general dips of the terranes comprising this part of the Ozark dome.

Compared with other portions of the Ozark region, the central Missouri district presents relief peculiarities which are not met with elsewhere, and which have a fundamental influence on the localization of the ore-bodies. The Osage river in its course washes the northwestern base of the dome. As shown by Davis,[42] this stream with its broad meanders has sunk deeply into the country from the position it occupied in the Tertiary peneplain before the uplifting of the Ozark region took place.

Since the Late Tertiary period the stream has been occupied not so much in laterally enlarging its valley as it has in cutting it downwards as the region was rising. The tributaries of the Osage have been doing likewise. Ground-water level has lowered several hundreds of feet beneath what it normally would be were the Osage river absent. Impounding of underground waters near the surface is now impossible. Whatever ore-bodies once existed in this upper zone have been largely removed. This is shown to some extent by the myriads of caverns throughout the region. While many minor ore-bodies still remain in this upper zone, the extensive ore-bodies of the central district must be sought at deeper levels than in other parts of the Ozark region—at, or below, the present deep-lying permanent water-level. It is doubtless on account of this fact more than any other that shallow prospecting has not proved so successful here as elsewhere in the region.

---

[42] *Science*, vol. xxi., No. 534, p. 226 (Apr. 28, 1893).

3. *Deposits of the Southwestern Missouri District.*—It is now a well-determined fact that developed ore-bodies of the southwestern Missouri mining-district all lie within the Springfield-Joplin syncline. The grouping of the mines in the various camps, and other features, suggest that there is probably a system of minor folds within the large one that has an important influence in separating the more-productive from the more-barren tracts. In other words, the great syncline referred to is in reality a compound trough.

The nearest suggestion to the Joplin trough being a true syncline is the observation of Winslow,[43] in which he accounts for the long Springfield tongue of Early Carboniferous rocks extending into the center of the Ozark dome from the Kansas line at Galena as an ancient estuary. He lays special stress upon this as offering an adequate explanation for the original

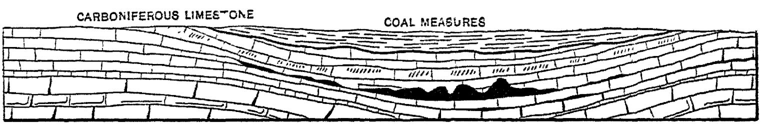


FIG. 9.—POSITION OF ORE-BODIES IN BOTTOM OF PITCHING SYNCLINE OF JOPLIN, MO.

concentrations of the ore-materials while the sediments were being laid down.

Cedar gap, Springfield, Ash Grove, Aurora, Granby, Carthage, Cartersville, Webb City, Joplin, and Galena all lie in or are located near the bottom of the syncline. In the immediate neighborhood of Joplin there appears to be a remarkable series of structural ridges trending at an angle of about 50° to the axis of the great syncline. These "cross-bars" have long been recognized by those familiar with the district. They are described in detail elsewhere. A significant fact connected with them is that the chief mining-camps are all located on the northeast and higher side of these cross ridges and in the pitching Joplin syncline.

The position of the ore-bodies with reference to the axes of the pitching syncline, at the foot of the slope from which the Coal Measures have been denuded, is indicated in Fig. 9.

4. *Deposits of Northern Arkansas District.*—In northern Ar-

[43] *Missouri Geological Survey,* vol. vii., p. 485 (1894).

kansas, as the arched strata of the Ozarks pass into the more closely folded structures of the Ouachita portion of the uplift, faulting becomes more frequent and more extensive than in Missouri. While it is manifest that most of the dislocations took place prior to the deposition of the ore-bodies in their present attitude, it is quite remarkable that there is so little direct association of the two. This is well shown by the results of the extended and special investigations of Hopkins;[44] and it is also in perfect accord with the later critical observations.

Although Branner's deductions appear to be largely hypothetical and to represent his general impressions of the field, it is manifest that he attaches considerable importance to shallow synclines as lines favorable to the location and development of new zinc-deposits.[45] There are, however, several localities where definite synclinal structures are now known to contain extensive ore-deposits. These features are developed notably

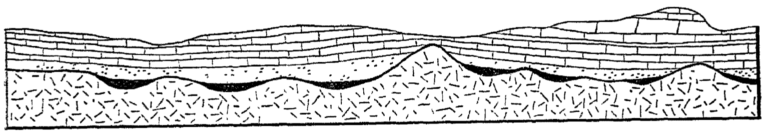


FIG. 10.—DISSEMINATED LEAD-ORES IN UNCONFORMITY-TROUGHS IN SOUTHEASTERN MISSOURI.

in the Sugar Orchard valley, south of Lead hill in Boone county, in the vicinity of Maryhattiana, and at some other points.

5. *Deposits of Southeastern Missouri District.*—The troughs throughout this part of the Missouri mining-field are mainly broad erosion-basins excavated in the Azoic granites. They are now completely filled with Cambrian sediments. Some of these old valleys, as shown by drill-holes in the vicinity of Pilot knob, are more than a thousand feet deep. They are well displayed, Fig. 10, in the vicinity of Doe Run, Mine La Motte, and other mining-camps nearby.[46]

Late investigations suggest that most of the larger ore-deposits bear a definite relationship to these anciently-formed structures. Indeed, some of the most successful prospecting has been done along the bottoms of the old basins or troughs. The conditions for the localization of ore-materials are, of

[44] *Arkansas Geological Survey, Annual Report* 1892, vol. v., p. 39 (1900)

[45] *Ibid.*, p. 32. [46] *Missouri Geological Survey*, vol. ix., pt. iv., p. 30 (1896).

course, essentially the same as in the case of pitching synclines. For all practical purposes the two structures are identical. The latter are also, probably, somewhat influential in the same region. Although no detailed investigations have been yet made leading to the association of ore-bodies with true synclines, there have been a sufficient number of the latter already noted to show conclusively that the entire country is gently flexed. Practical prospecting suggests that the synclines, as well as the erosion-basins, should be located and carefully examined for the more extensive ore-bodies.

6. *Synclinal Ore-Deposits of Other Districts.*—The accumulations of ore-materials into large ore-bodies in structural synclines strictly within the vadose zone have been noted in a number of localities, although until quite recently their practical significance had not been fully appreciated. The famous Lake

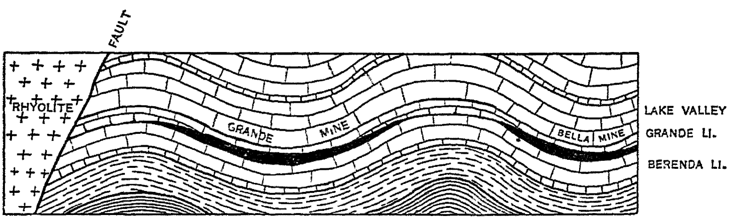


FIG. 11.—ORE-BODIES IN SYNCLINES AT LAKE VALLEY, N. M.

Valley (New Mexico) silver-deposits have been shown[47] to lie wholly within structural troughs, as shown in Fig. 11. All of the ore-bodies have been very definitely associated with the sharper corrugations of the older stratified (Carboniferous) rocks in which they occur. The ore-materials have been segregated in the shallow synclines, which now pitch at an angle of about 20°. It is believed that these pitching troughs were primarily involved in the accumulation of the ores illustrated in Fig. 12.

In the Wisconsin lead-region, Chamberlin[48] early noted the location of mines in structural troughs rather than on anticlines. The details of the proofs that this association is really a genetic one were more recently presented by Grant.[49] Other instances might be mentioned. The phenomenon is one that

---

[47] *Trans.*, xxxix., 139 (1909).

[48] *Geology of Wisconsin*, 1873 to 1879, vol. iv., p. 432 (1882).

[49] *Economic Geology*, vol. i., No. 3, p. 240 (Dec.-Jan., 1906).

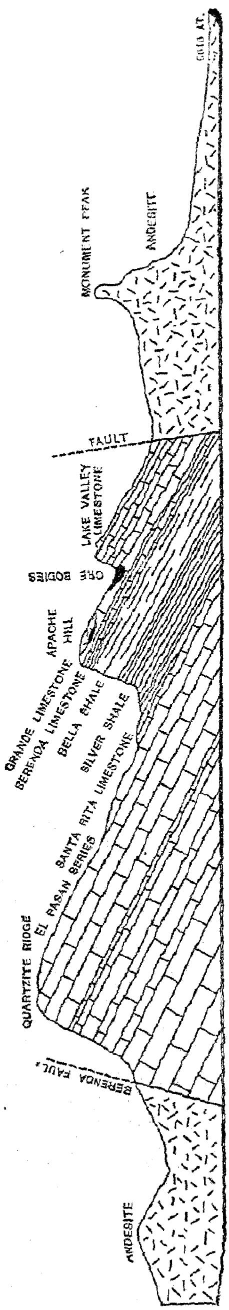


FIG. 12.—LONGITUDINAL SECTION OF PITCHING ORE-BODY AT LAKE VALLEY, N. M.

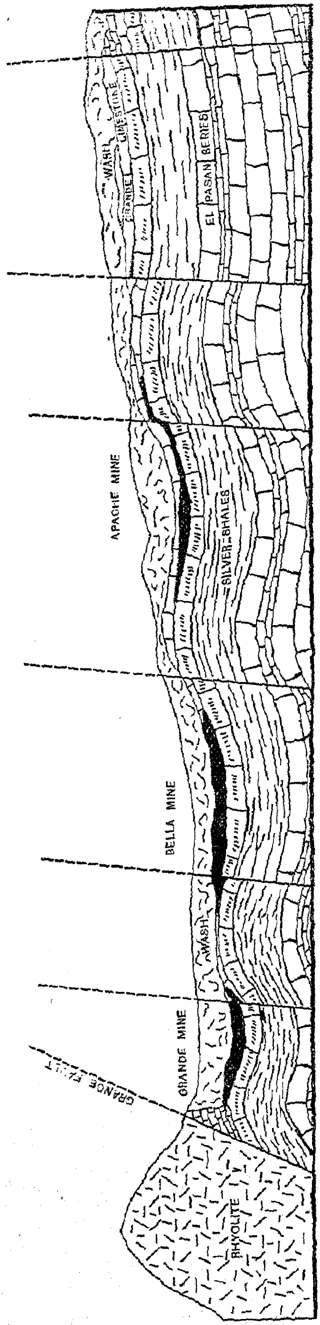


FIG. 15.—ORE-BODIES IN PITCHING SYNCLINE AT LAKE VALLEY, N. M.

is doubtless much more prevalent than is commonly supposed. Its importance was recognized in the scheme of a classification of ore-deposits upon a geologic basis by making this manner of ore-occurrence one of the general types.[50]

7. *Significance of Ore-Troughs in Prospecting.*—In strictly vadose depositions, the localization of the more productive ore-deposits in areas which are occupied by more or less well-defined basins or synclines, appears sufficiently well established to make the suggestion of great practical value in prospecting new mineral ground and in encouraging further systematic exploration of old mining-camps. In the Ozark region these geologic conditions seem to obtain with remarkable clearness. There are, to be sure, many apparent exceptions, but attention

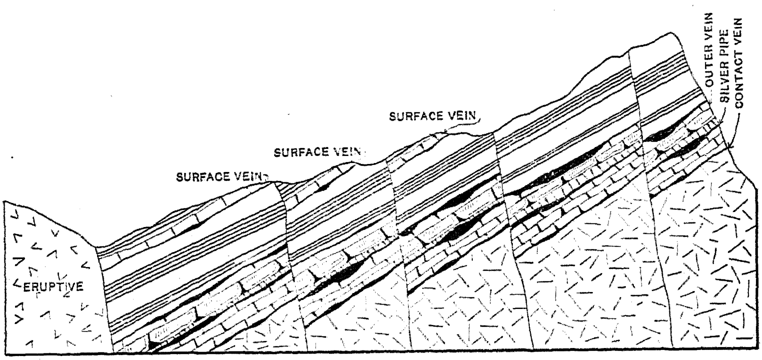


FIG. 13.—IMPOUNDED ORE-BODIES AND FAULTING AT MAGDALENA, N. M.

is called to some of the modifying factors in the reference to the origin of minor synclines or basins.

Under certain circumstances repeated faulting may give rise to similar conditions so far as concerns the tendency to impound ore-bearing waters, as has been described in the Magdalena zinc-district of New Mexico;[51] but in the Ozark region the same conditions are imposed by folding. It is noteworthy that in the Magdalena ore-bodies there is a local water-level in each fault-block, and that above this level in each case the ores are chiefly carbonates, while below this level the ores are sulphides. What is more remarkable, the fault-blocks are only 200 or 300 ft. across in some instances, as illustrated in Fig. 13.

[50] *Trans.*, xxx., 348 (1900).

[51] *Mining Magazine*, vol. xii., No. 2, p. 109 (Aug., 1905).

The recognition of sharply-defined basins, or troughs, in all of the great mining-districts of the Ozark region; the apparent occurrence of the majority, at least, of the most productive mines in these basins; the determination of the existence of similar conditions in other mining-regions where the ores are strictly vadose accumulations; and the great theoretical probability of the genetic relationship between the troughs and the ore-bodies, clearly point out the direction of most fruitful prospecting for new ore-deposits and new fields.

The careful determination of minor synclines around the marginal zone of the Ozark dome would probably promote the mining-industry of the region many fold more in a few months than the present manner of desultory drilling and prospecting can in a hundred years. Judging from general conditions, the extent and character of present mining-operations and the relation of the ore-bodies to the geologic structures, suggests that only a very small part of the ore-deposits of the Ozarks has as yet been discovered.

8. *Origin of the So-called Synclines.*—As presented in the geologic cross-sections, any sagging of strata constitutes a syncline. Flexing of this kind is usually, without further question, ascribed to mechanical deformation of the same class as that to which mountain-building belongs. This, however, is not always true.

Geometrically, the phenomenon is considered only in two dimensions. It is really a problem of three dimensions. As such, the form to be considered is best regarded not as a simple syncline, or even as an irregular trough, but as a basin. So far as ores are concerned, synclines may be treated broadly as basins and their origin not restricted alone to those structures which are formed when rock-masses are under great lateral compression.

In their broader relationships minor "synclines," or basins, may be regarded as formed (*a*) through original unevenness of the local sea-bottom, (*b*) through irregularity in deposition of the sediments, (*c*) through flexing of strata on account of orogenic movements, (*d*) through bending of layers during the process of faulting, (*e*) through settling of strata on account of local changes in volume of terranes beneath, loss of gases, solution, or metamorphism, (*f*) through local intrusions of

igneous rocks, and (*g*) through general tilting of already uneven strata, as along unconformity-planes.

Basins formed through the original unevenness of the sea-bottom are most easily determined when that part of the old sea-floor was previously a land-surface subject to erosive influences. A most noteworthy example is that of the southeastern Missouri mining-district. As shown by drill-records, pre-Cambrian valleys, from 1,000 to 1,500 ft. deep, are not uncommon. While recent erosion has exhumed the ancient crystalline mountains of St. Francois, and has already removed most of the sedimentaries which once covered all of the old peaks, the old valleys still contain Cambrian limestones and sandstones several hundreds of feet thick. Many of the principal ore-bodies now lie near the bottoms of these ancient drainage-troughs. It seems probable that most, if not all, of the ore-deposits of the district will be eventually found to have some direct connection with the courses of the old troughs. Observations bearing upon this very point were published nearly 15 years ago,[52] and since that time practical tests have fully confirmed the original working-hypothesis.

Basins formed by irregularities in deposition of the sedimentaries appear to be relatively unimportant in the Ozark region; although they are known to be influential in ore-deposition in other mining-districts. However, some of the lesser deposits of the southeastern Missouri district seem to be directly referable to this class.

Of late years much has been done towards locating true synclinal basins, and towards the measurement of their extent and amplitude, in ore-producing areas. So far as present observations go, flexing has occurred throughout this region at many different times and in different directions. The synclines bearing immediately upon the segregation of the ore-bodies are those which pitch down the slope of the great dome. The supposition would be that the chief mining-districts should display noticeable synclinal structures. With this expectation the facts accord. In the southeastern district, marked, though gentle, flexing characterizes the strata. It seems probable that these structural troughs have influenced ore-deposition quite

---

[52] *Missouri Geological Survey*, vol. ix., pt. iv., p. 58 (1896).

as much, perhaps, as the basins caused by the inequalities of the pre-Cambrian erosion-surface. The system of true synclines pitching northwestward in the central Missouri district has already been described. Synclinal structures of the northern Arkansas district also have been noted; as have been those of the Joplin area. Special attention is directed later in this paper to some of the peculiarities of the last-mentioned district.

The bending of strata near lines of faulting is not met with in the Ozark region, so far as known, although it is a frequently-occurring phenomenon in other mountainous areas.[53]

The unequal settling of strata is a geological phenomenon of universal extent. It is one of the constant causes of irregularities in stratification. Change in volume of underlying beds from any cause gives rise to local basins, troughs, or synclines. As an important factor in prospecting, the verity and wide extent of the phenomenon were first impressed in connection with investigations of the coal-deposits of Iowa.[54] In working out the detailed stratigraphy of the Des Moines district, it was found that, irrespective of the general depression of coal-beds at the center, there were other irregularities of basin-like character to be accounted for. The inference, then, was that the basins in one coal-seam might be made use of in locating the central thicker parts of lower coal-beds by drilling in the middle of these sags in the coal-seams above already opened.

By careful surveys of these coal-seams of the central Iowa region, and special examinations in other parts of the Iowa and Missouri coal-fields, Bain[55] afterwards proved that such relationships not only actually did exist, but that they were of wide extent. Subsequently, the same author accounted for many of the local sags in strata of the Missouri lead-districts in the same way. It is also referred to by Grant in Wisconsin.[56]

Similar local sags in strata undoubtedly are of very frequent occurrence in such regions as the Ozark uplift, where the irregular shrinkage of rock-masses, due to dolomitization and like chemical alterations, is known to be extensive. While exact determinations regarding the local settling of strata might

---

[53] *Engineering and Mining Journal*, vol. lxxviii., No. 17, p. 670 (Oct. 27, 1904).
[54] *Iowa Geological Survey*, vol. ii., p. 279 (1894).
[55] *Journal of Geology*, vol. iii., No. 6, p. 646 (Sept.-Oct., 1895).
[56] *Economic Geology*, vol. i., No. 3, p. 240 (Dec.-Jan., 1906).

be difficult to make in particular instances, and while other and more conspicuous structures might, and probably often do, obscure those of this class, it is a factor that should receive the fullest consideration in prospecting for ores throughout the region.

The bending of strata on account of the effects of local intrusion of igneous masses is, in the region under consideration, not so influential in localizing ore-deposits as in some other parts of the world. In this connection it may be neglected altogether. Only two points are at present known where such disturbances have taken place. One is in Camden county, Mo., first made known by Shepard, and described by Winslow,[57] and the other is on Spavinaw creek, in Oklahoma, near the Arkansas border, described by Drake.[58]

The tilting of planes of uneven unconformity through orogenic movements is productive of new basin-conditions. To what extent this has operated in the Ozark area, where the inclination of the strata is everywhere comparatively slight, is not, at present writing, known. Some of the smaller deposits of the Joplin district appear to be explicable upon no other supposition.

9. *Significance of the Cross-Bars of Joplin.*—For many years it has been noted that, in the vicinity of Joplin, especially, the most striking inclinations of strata often have a trend in a NW–SE. direction. Among the miners, who are most familiar with the local structural conditions of the area, it is agreed usually that the Ozark uplifting could have had no effect upon the geologic structure because the sharper bendings of strata strike northwest instead of northeast.

The idea is given even greater prominence by the recent publication of Smith and Siebenthal,[59] who recognize a single stratigraphic arch, having an amplitude of less than 150 ft. On account of its passing near the city of Joplin it is designated the Joplin anticline. In the field, this arch, which has long been recognized by the miners, is more pronounced than might be supposed. By the authors just mentioned, attention is especially called to the course of the arch as represented on

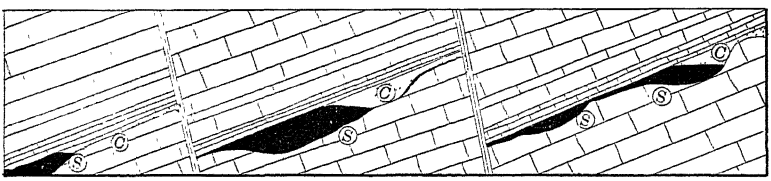
57 *Missouri Geological Survey*, vol. vii., p. 432 (1894).

58 *Proceedings of the American Philosophical Society*, vol. xxxvi., p. 338 (1897).

59 *Geologic Atlas of the United States*, Folio No. 148, p. 9 (1907).

their underground-contour scheme of the surface of the Grand Falls chert-terrane.

Within the area covered by the map in question there are really no less than four of these anticlines, all parallel to one another. While they are all nearly as prominent as the Joplin arch, they are not so well represented by the underground-contours of the map. The towns of Galena, Joplin, Cartersville, and Carthage lie on the eastern slopes of their respective arches. There are other arches of the same series beyond the limits of the mapped area. Significantly, the main ore-deposits also lie almost wholly on the east side of the structural ridges. While the great Joplin-Springfield syncline pitches westward and radially down the slope of the Ozark dome, the cross-bars, or arches, act somewhat as do dams placed at regular intervals in streams. The vadose waters are partly impounded. A



C, carbonate. *S*, sulphide.

FIG. 14.—STRUCTURE OF ORE-BODIES AT MAGDALENA, N. M.

model of the structures would be not unlike the succession of riffles in a gold-washing sluice-way.

Tectonically considered, the Joplin cross-bars were manifestly formed at a period far removed from that in which the larger Joplin-Springfield flexure originated. The latter I am inclined to associate with the system of foldings, now so apparent in the Boston and the Ouachita mountains to the south, before the upraising of the present Ozark dome. The former seem to belong to a much earlier system. They are themselves flexed.

A similar system of cross-bars, produced, however, by small transverse faults, but giving identical impounding conditions to a marked degree in a well-defined and steeply-pitching syncline, is shown in the Magdalena zinc-district[60] of New Mexico, a section of which is represented in Fig. 14.

[60] *Mining Magazine*, vol. xii., No. 2, p. 109 (Aug., 1905).

10. *Relation of Ore-Run to Ancient Relief-Features.*—Miners in Missouri are prone to fancy some intimate connection between the present stream-valleys and the occurrence of ore-runs. In a region where typical Karst topography prevails, where much of the drainage is underground, where sink-holes are frequent, where caves abound, and the country is a veritable land of springs, the subterranean water-ways have determined, in a way, many of the local lineaments. The direct association of the two has been recently pointed out by Clerc.[61]

On the other hand, some writers, as Buckley and Buehler,[62] have attempted to localize the ore-deposits of the Granby area and the southwestern district generally along lines of ancient drainage-ways, ascribing the development of the ramifying channels to the period when the erosion-plane, represented by the unconformity at the base of the Coal Measures of the region, was formed. The evidences adduced in support of this hypothesis are far from convincing. The geologic history of the region is strongly against such supposition.

To begin with, general planation or peneplanation was, during the great Arkansan period of Mid-Carboniferous times,[63] very complete over the entire area of the present northern Ozarks. This surface was, according to all evidences, only slightly elevated above the sea-level. There are no proofs whatever of marked stream-ways, much less of deep gorges in the bed-rock. This appears to be true of the entire region from the Arkansas line to Minnesota. The channel-ways in the upper eroded surface of the Early Carboniferous limestones, now filled with sandstones and shales—the so-called Coal Measures débris—appear in every observed case to be the result of accumulations of much later date.

In the case of most of these isolated deposits of sandstones and shales, there has been undoubtedly a filling-in by the caving-in of the roofs of underground drainage-ways and lines of caverns which were excavated in the soluble limestones immediately beneath the basal beds of the Coal Measures. So far as personal examinations go, it seems quite manifest that many, if

---

[61] *Trans.*, xxxviii., 320 (1908).

[62] *Missouri Bureau of Geology and Mines*, vol. iv., p. 61 (1906).

[63] *Proceedings of the Iowa Academy of Sciences*, vol. vii., p. 123 (1901); also, *Bulletin of the Geological Society of America*, vol. xii., p. 173 (1900).

not all, of the described "disturbed" accumulations of this region are of like origin, Their formation could not, therefore, have been in Carboniferous times, but at a much later date, after the uprising of the present Ozark dome, and while the easily-removed shales of the Coal Measures were being peeled off the inclined surface of the limestones. Instead of being the means of localizing ores, these "Coal Measures" are merely accidental associations. They are formed in the last stages of underground stream-ways, when the greater part of the ore-materials had already been removed to lower levels, or exported beyond the limits of the district.

There are, of course, outliers of the true Coal Measures now resting in shallow basins upon the old eroded surface of the limestones, but these do not appear to be commonly associated with notable ore-bodies. It is the gorge-sands and clays that are considered as fillings from sink-holes or caverns, the materials in many cases being derived, in great part perhaps, from Coal Measures débris. This phenomenon appears not uncommon wherever the Karst topography prevails in the Ozark region. It is described by Nason[64] in explanation of the origin of the iron-ores of the region.

IX. Direction of Ground-Water Circulation.

The surface-drainage of the Ozark dome is, in the main, radially down the slopes. For reasons already mentioned, the general movement of the phreatic waters is likewise principally in the same direction. Everywhere around the uplift the general dip of the rock-strata is at somewhat higher angles than the general surface-inclinations. Nevertheless, the ground-water level reaches day-light at many points.

The general artesian conditions of the region are peculiar. With a head of from 700 to 800 ft., ideally favorable structural conditions, extensive porous beds, and broad catchment-areas, the supposition is that artesian flows would be encountered wherever drillings might be made in proper locations. Such expectations are, however, not realized. Shepard[65] has recently conclusively shown that the artesian conditions are not

[64] *Missouri Geological Survey*, vol. ii., p. 134 (1892).

[65] *Water Supply and Irrigation Paper* No. 195, *U. S. Geological Survey* (1907).

what they ordinarily should be in a region of the character of the Ozarks.

With rock-beds pitching so nearly with the topographic inclination, with soluble rock-layers at regular intervals, with a remarkable system of jointing, and with extensive, ramifying underground water-ways at small depths, it would seem that, all things considered, the main water-circulation would be in the vadose belt, and very nearly parallel to the general slope of the dome. This movement of the subterranean waters near the surface must be relatively so rapid and the current so copious that any deeper circulation would be, in comparison, of little importance.

The extent of an ascending artesian circulation, or general upward movement of mineral-bearing waters, as urged by Van Hise and Bain, is certainly very greatly overestimated. The hypothesis appears to find small support in direct evidence of a critical character. It is, of course, theoretically possible for an upward circulation from the Ordovician horizons to exist, whether or not the region be profoundly faulted. Without any faulting at all, the rocks both above and below the basal Carboniferous shales, as well as the shales themselves, are brittle enough and jointed enough to permit the free passage of ample volumes of artesian waters. But it is on account of this very fact that the chief movement of the subterranean waters is near the surface, closely following the stratification-planes, and not mainly a deeper circulation that ordinarily would be expected to obtain.

The question of the presence or absence of marked fault-planes does not affect by one iota Bain's contention of deeper ascending currents. The artesian circulations of Iowa, by way of comparison, do not appear to be metal-carrying to any appreciable extent; and the artesian circulations of the Ozark region, having identical conditions, even to the extent and character of the lower or basal aquifers, are not necessarily more important in this regard.

Were the deeper artesian waters the most mobile, and were they the main carriers of the ore-materials, the fault-lines would be naturally the principal lines of ore-deposition. But the most recent observations go to show rather conclusively that the fault-lines are not usually *loci* of ore-deposits; and the evi-

dences on this point are now so incontestable that Bain himself has been obliged of late[66] to admit their force. It would seem, on the whole, that on this one point alone rests the whole proof of the truth or falsity of the Van Hise theory. If this theory were susceptible of proof at all, the Ozark region above all others in the world is the one place where it can be critically tested.

In the radially-pitching synclines of minor character, to which special attention has been already directed, the vadose waters naturally have a tendency to work towards the middle of the troughs. The general surface of the Springfield-Joplin syncline, for instance, is a stratum-plane from the crest of the dome, at Cedar gap, down to the base of the slope, at the Kansas boundary. Unusual conditions being absent, the greater volumes of ground-waters should move down the center of the syncline. This being the case, and the proper impounding conditions being present, the main ore-bodies should accumulate chiefly along the middle line. Such appears to be the fact.

X. Vertical Range of Workable Ore-Bodies.

The significance of shallow mining in areas so extensively developed as are those of the Ozarks, of the absence of marked faulting, of the negative evidences of deep prospecting, of the lateral expansion of the ore-bodies, and of the general geologic features, does not appear to have been duly weighed by those observers who have tried to account for the deep-seated origin of the ores.

As is now well known by all who are engaged in mining in the region, few deposits are found below the level of about 250 ft. from the surface. In southwestern Missouri little ore appears to occur lower than depths of from 200 to 250 ft. This is the well-marked horizon of the base of the Grand Falls chert-formation which occupies the middle section of the great Carboniferous limestone sequence. In the southeastern district the same is generally true, although there is no distinctive horizon beneath which the ores are limited, except, of course, when the great unconformity-plane at the base of the Cambrian section is approached. While the deposits of the central district at

---

[66] *Economic Geology*, vol. i., No. 2, p. 172 (Nov.-Dec., 1905).

present opened are still nearer the surface, there is good reason to believe that the more extensive ore-bodies are to be sought at greater depths.

Sulphide ores, in scattered particles and thin laminæ, are found throughout the limestones of the Ozark region, as well as far beyond its borders. They doubtless, also, occur in the depths, even to the crystalline complex. Thus far both actual drillings and theoretical considerations give no encouragement to search systematically for deep ore-deposits. All evidence suggests that the main efforts in prospecting for new ore-bodies should be confined to the principal enrichment-zone extending downwards a distance of 100 ft. or so beneath present ground-water level.

The Ozark ore-deposits in general are thus to be regarded as strictly vadose accumulations.

XI. Oscillations of Ground-Water Level.

There are certain features connected with the Ozark ore-deposits, formed through recent and considerable changes in the level of ground-water, that have long been very puzzling. The phenomena have not been generally recognized either in this or other mining-districts. In the vadose ore-zones they are factors of very considerable practical importance. The recognition of recent and distinct lowerings or risings of the local ground-water levels may alter the entire plan of mining-operations.

Throughout the region the conditions are such that the local oscillations of the ground-water level may readily and frequently occur. The sudden damming of a large subterranean drainage-way may elevate the permanent water-level of the locality to the extent of 200 and even 300 ft. Thus, oxidized ores are often found considerable distances below the level where they would be normally expected to be limited. On the other hand, the relatively rapid cutting-away of cross-barriers of impoundment might soon lower the local water-level, leaving sulphide ore-bodies high above their ordinary positions. The same phenomena are produced much more slowly, perhaps, by faulting, flexing, and warping.

In many cases the immediate causes of ground-water oscillations are distinguishable. The peculiarities of the Osage river

erosion already have been mentioned in this connection. The Neosho and White River drainages afford other good examples. In all of these the details of the problem of the local changes of ground-water level are as full of scientific interest as they are of practical significance. It is, on the whole, a subject that deserves the most careful consideration in all mining-operations throughout the region.

XII. Relation of Ore-Bodies to Lithoclasic Features.

The burden of evidence at the present time seems to be that in general there are no direct relationships between the ore-bodies and fault-lines. The location of the latter holds out no hope for the discovery of any extensive deposits of the former in connection with them. This is the main point to be emphasized. The proofs have been discussed at some length in previous pages.

XIII. Primary Source of Ore-Materials.

In the various discussions on the ore-deposits entirely too much stress is placed upon the relations of the ore-bodies to the primary sources of the ore-materials. This phase of the subject is really of small importance either practically or scientifically. It is not only possible but certain that there are sufficient ore-materials finely disseminated through the rocks of the Palæozoic section of the region to supply ore-bodies hundreds of times greater than those now known. There need be little contention over the point of ample supplies.

It is not necessary to go back, for an adequate source of the present ore-bodies, to the possible but slight concentrations by ocean-currents at the time when the sedimentaries were laid down. The present influence of such factors, if it ever was an appreciable reality, is practically *nil*. Besides, even if such effects were produced by the Cambrian sea, subsequent changes in the rock-masses and the general underground-water circulations have long since obliterated every trace of accumulations of this kind. The theme at best now has only a vague speculative interest.

The primary source of the ore-materials of the region cannot be considered as the old crystalline rocks, of which the granites of the St. Francois mountains are types, eroded and weathered

in pre-Cambrian times, and now exhumed again. The débris carried away from these old hills was transported far beyond the boundaries of the present Ozark dome. Those sediments now covering the St. Francois mountains were derived from very distant areas. The diffused metallic content of the Ozark rocks cannot, therefore, be regarded as derived in any way from the old crystalline basal complex now exposed in the region.

In this connection, the concentration of ore-deposits through great and long-continued surface-decay of rocks, in which ore-materials have been retained in a state of diffusion ever since the time when the rocks were first consolidated, should be noted. This is Winslow's [67] hypothesis. It is essentially the theory of the evolution of fissure-veins, applied to a large area. It has been rather specifically shown [68] that this extension of the idea is entirely untenable. It alone does not account for concentration of the diffused ore-materials into workable deposits. The latter require, in addition to a movement of ore-particles, geologic conditions by which they come to rest in restricted areas or belts.

The liberation, through general surface-weathering, of ore-materials held in the upper rocks is an inadequate explanation for the diffusion of the metallic particles through the lower rocks. There are three geologic periods during which there was notable planation of the region. The first of these great erosion-periods occurred early in the Mid-Carboniferous time, the second in Early Tertiary time, and the third in Recent time. It requires no argument to defend the statement that the process was sufficient during any one of these three periods to diffuse ore-materials throughout the entire rock-mass from the surface down to the crystalline basement. Besides, enormous amounts of ore-materials must have been carried out of the region, as they are to-day, in the surface drainage.

Recent chemical analyses of rocks of the Upper Mississippi valley by Robertson,[69] Weems,[70] and others, clearly show that appreciable amounts of the common metals exist in nearly all rocks. In some areas, later general enrichment may occur

---

[67] *Missouri Geological Survey*, vol. vii., p. 477 (1894).

[68] *American Geologist*, vol. xxvii., No. 6, p. 361 (June, 1901).

[69] *Missouri Geological Survey*, vol. vii., p. 479 (1894).

[70] *Iowa Geological Survey*, vol. x., p. 567 (1900).

from time to time, until the amount of diffused ore-material reaches a notable fraction of 1 per cent. In other localities actual impoverishment may go on. The central part of the Ozark dome appears to be such an area.

The essential factor to be taken into consideration is that there may take place at any time, whenever a region becomes more or less elevated above the sea-level, a wide-spread diffusion of ore-materials, amply sufficient for concentration subsequently into the largest of ore-bodies whenever the geologic conditions are favorable.

XIV. Zone of Ore-Enrichment.

If we could rightly compare the Ozark ore-bodies with those of normal fissure-veins with their three distinct zones, they would represent only the middle zone, or zone of enrichment. The upper zone of general impoverishment, or oxidation, and the lower zone of normal low-grade sulphides, are both wanting. At least, they are so poorly developed as to be practically absent. This apparent anomaly is the result of peculiar, though not uncommon, geologic conditions. The latter are conditions which are not commonly taken into account. The general movement of the altering ore-body, instead of being directly downward upon itself, is directed laterally along the pitching stratification-planes.

In the ordinary sense in which the term is used in describing fissure-veins, there is, among the Ozark ore-deposits, really no zone of enrichment. It is a case of direct deposition of sulphide ores under unusual local conditions in what has until recently been the weathered zone. The local conditions are essentially the same as they are in the normal zone of enrichment in fissure-veins. There is, at local ground-water level, an impounding of the waters ladened with metallic salts. Hence, the recognition of the likenesses and differences of conditions under which the two kinds of ore-bodies are found is of the greatest practical value.

These fundamental peculiarities explain why, in the Ozark region, there is such a marked absence of ordinary oxide ores above the sulphide-zone, and why, below the "zone of enrichment," which is usually the only level mined, we should not expect to find a great zone of workable low-grade ore. The

lower limit of the zone of enrichment is nowhere probably a particular geologic horizon, as, for instance, the base of the Grand Falls chert in the Joplin district, but a plane approximately parallel to the general surface of the ground, and everywhere beveling the local stratification-planes at a low angle. This nethermost limit may be, in the western part of the Joplin district, from 100 to 200 ft. above the basal horizon of the chert terrane, while in the eastern portion of the same district it may be as far beneath the same stratigraphic level.

The basal limitation of present ore-deposition is probably nowhere more than 300 ft. below the general surface of the Ozark dome. This does not preclude the possibility of the existence of important ore-bodies beneath. The depths at which ore-bodies occur are, therefore, not genetically limited by definite stratigraphic planes, but are to be measured from the general surface-contours of the region.

XV. Geologic Age of Present Ore-Deposits.

As shown recently,[71] ore-deposition in general has not been confined to a single geologic period, but has been in active operation at very different times. The periods of active ore-formation are closely associated with the orogenic activities of the region. When these are important the ore-depositions are important; when there is orogenic quiescence there are practically no important ore-bodies localized. Orogenic movements being the main factors indicating the inauguration of new conditions that are favorable to the local concentration of ore-materials, the cycles of physiographic development have a direct bearing upon ore-genesis. In physiography, therefore, geologists have come into possession of an extremely delicate means of determining relative earth-movements. In the region occupied by the Ozark highland this test has been proved to give exceptionally exact values in fixing the time-relations of the ore-deposits.

The last period of vigorous planation has been shown, upon abundant evidences, to be in Late Tertiary times, just before which period the entire region had been parallel-folded from compression, acting in a north and south direction, and upraised. The country was eroded down to near sea-level, exposing all

[71] *Trans.*, xxxi., 603 (1901).

of the stratified rock-section to the crystalline basement. In the middle of the region the Early and Mid-Carboniferous sections, and probably also the Cretaceous section, which once extended entirely over the area, were removed. There does not appear to be the slightest reason for believing that this region has been from early Cambrian times exposed to atmospheric agencies. The myth of the Ozark isle should no longer find countenance in the geologic history of the region.[72]

In the Ozarks, ore-deposition certainly began to proceed vigorously with the commencement of the last period of uplifting of the region, which occurred since Tertiary times. The uprising is even now going on rapidly—as fast, perhaps, as mountain-building ever does go on, and with it ore-formation also. The present localizations of ore-deposits, which include practically all of the bodies now worked extensively, are assigned a very recent geologic date. It is not at all unlikely that most, if not all, of the Ozark ore-deposits of this class were really formed within the memory of man.

XVI. Controlling Factors of Localization of Ore-Materials.

The general association of the workable ore-deposits of the Ozark region with basin-structures is far more significant in its bearing upon systematic prospecting than, at first glance, might appear. It seems to make small difference what is the immediate origin of these basins. They may be formed by true synclines in ordinary flexed strata, by old erosion-troughs along planes of unconformity, by warpings of impervious beds, by cross-folds in structural troughs, by solution-cavities or channels, by underground water-ways dammed by débris, by faulting, by dikes or intrusions of igneous rock, by lack of erosion of impervious beds in sloping strata, or by other means. The principal point to be emphasized is that the introduction of these structures gives rise to an impounding of the subterranean waters with their mineral content in solution, and a production, on new lines, of all of the conditions of a new and permanent ground-water level.

Sulphatic waters, passing into and through such subterranean ponds, drop some of their metallic loads in the form of sul-

[72] *Science*, N. S., vol. vii., No. 174, p. 588 (Apr. 29, 1898).

phides, just as they do in the enrichment-zone of normal fissure-veins. By what means the precipitation takes place is, in the present connection, immaterial. It may be through the mingling of different currents, through the presence of organic matter, through the association of carbonaceous material, or through the presence of decomposing sulphides. In this zone, and under impounding conditions, there is invariably present an abundance of reducing-agents.

In the southeastern Missouri lead-district the localization of the ore-bodies appears to be chiefly influenced by the character of the pre-Cambrian channel-ways corraded out of the still older granites. The central Missouri district presents true structural troughs or synclines as the controlling feature, although subterranean channel-ways, now filled with sandstones and bituminous shales or clays, have occasioned minor ore-bodies. Both synclines and faults have produced basin conditions in the northern Arkansas district. In the Upper Mississippi district synclines alone are determining factors.

Conditions are complex in the southwestern Missouri zinc-district. That they appear more complicated in this area than they do in other districts may be due partly to the fact that this area has received very much more detailed study than any of the others. The great Springfield-Joplin syncline, pitching westward down the slope of the Ozark dome, clearly explains many of the former puzzling features connected with the distribution of the ore-bodies. The cross-bars, or arches, long known in the vicinities of Galena, Joplin, Webb City, Carthage, and other places, are believed to be of first importance in the localization of the ores. Close-patterned warping of the strata has been suspected ever since the detailed, large-scale topographic maps of the Carthage, Joplin, and other neighboring quadrangles were first constructed by the Missouri Geological Survey, more than 15 years ago.

Although not so accurate as it might be in many ways, but still presenting the broader features, the late publication by the U. S. Geological Survey[73] showing the underground-contours of the Grand Falls chert-surface, displays the relations of the ore-bodies and the basins in a remarkable way. Ore-bodies in large solution-cavities, particularly along the major lines of

---

[73] *Geologic Atlas of the United States*, Folio No. 148 (1907).

jointing, and in dammed underground water-ways, are also of common occurrence in the Joplin district. Corrasion-channels in the plane of unconformity at the base of the Coal Measures do not, at present writing, appear to exist. So far as personally known, the débris, sandstones, and shales usually found in this connection are not depositions of Carboniferous times at all. They appear to be fillings of very recent underground channel-ways that have caved in or filled up, often without the roofs giving away. As such, there must be seldom any relationship between the pre-Coal Measures drainage-lines and the ore-bodies.

The Ozark region is one in which some of the most influential structural features mentioned are not at first very easily determined. The general dips of the strata are so small, the flexing so slight, the warping so close-patterned, that it has only been through the application of recent and refined methods of investigation that the actual proofs of some former deductions were secured. The delicacy of the standard of measurement may be judged when it is stated that a warping or flexing of less than a dozen feet may give rise to basins sufficient to contain the largest ore-bodies. The principles involved are, perhaps, best considered by referring first to examples of other mining-regions, where, with the same vertical values, the lateral components are infinitely smaller in comparison, and the angles much higher.

The silver-deposits of Lake Valley, New Mexico, which were recently described,[74] illustrate the case in hand. These deposits are near the surface. They were formed at horizons that until very lately were the local ground-water levels. While they are now mainly oxidized ores, some sulphides still remain. The ore-bodies are confined in the bottoms of pitching synclines, Fig. 15 (p. 208), of which there are a number closely parallel to one another. The relationships of the ore-bodies to the limestone escarpment, which impounds in a measure the ground-waters and the surface-waters also, are represented in longitudinal section in Fig. 16.

A still more remarkable example is found in the Magdalena (New Mexico) lead- and zinc-district.[75] Here repeated faulting

---

[74] *Trans.*, xxxix., 139 (1909).

[75] *Mining Magazine*, vol. xii., No. 2, p. 109 (Aug., 1905).

has brought impervious layers against more soluble beds. The general geologic structure of the Magdalena range is represented in cross-section in Fig. 17, the horizontal distance of

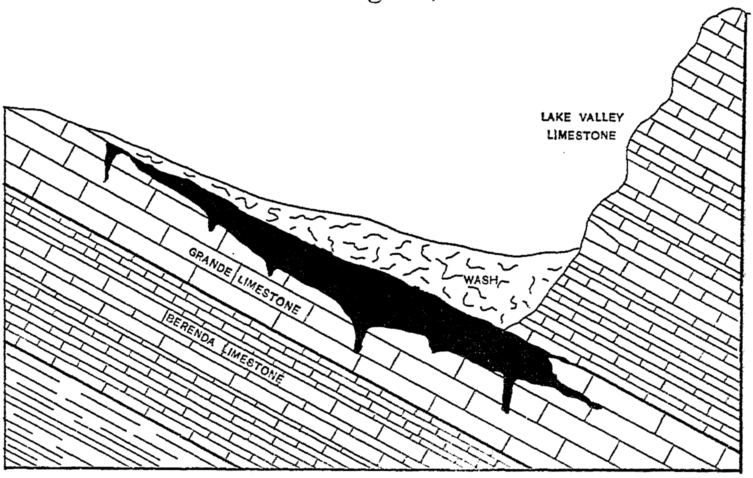


FIG. 16.—RELATIONS OF ORE-BODIES TO LIMESTONE ESCARPMENT (LAKE VALLEY, N. M.).

the section being about 20 miles and the vertical distance about 5,000 ft. The ores are deposited mainly along stratification-planes in the Early Carboniferous limestone now forming the crest and back-slope of the mountains. The details of the

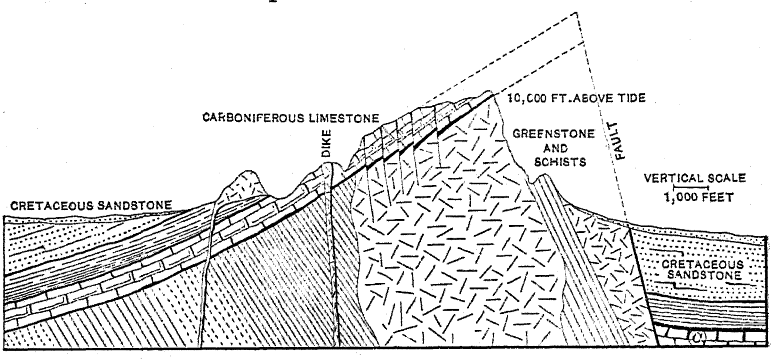


FIG. 17.—STRUCTURE OF MAGDALENA MOUNTAINS: ORE-BEARING SYNCLINE PITCHING WITH THE DIP (NEW MEXICO).

ore-bodies and the faulting are shown in Fig. 14. It is to be especially noted that in each fault-block there is a lower part of the ore-body (black) that is in sulphide form, and an upper part (dotted) that is the carbonate. The line separating the

two kinds of ore is a level that extends from the intersection of the ore-horizon with the fault-plane in the adjoining fault-block. This is the point of overflow, as it were, from the higher fault-block, and is the local ground-water level. It is beneath this level in each block that the water is in a condition of normal impoundment, and the conditions are essentially those which normally exist below ground-water level.

In the Joplin district the profile of the Grand Falls chert, along a line passing through the principal mining-camps, brings out several important features, shown in Fig. 18. The chief ore-bodies are located in basins, due partly to close-patterned warping and partly to parallel flexing. The ridges of Galena, Joplin, Webb City, and Carthage are true anticlines trending nearly northwest. They are the so-called cross-bars of the Spring-

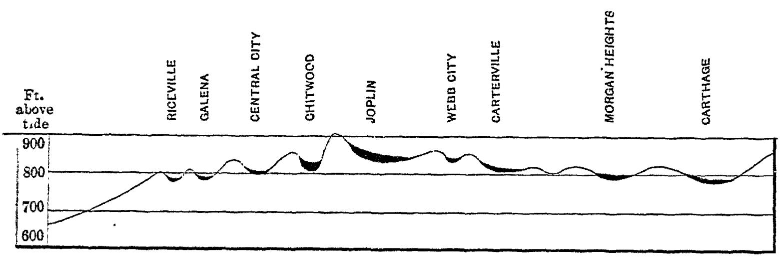


FIG. 18.—CROSS-BARS OF JOPLIN DISTRICT, SHOWING RELATIONS OF ORE-BODIES TO GEOLOGIC STRUCTURE.

field-Joplin syncline, which pitches west. The ore-bodies are thus seen to have definite relationships with the geologic structures. It seems very probable that the retreating margin of the impervious Coal Measures shales has had an important influence in impounding the vadose waters by raising the local ground-water level. There is, moreover, reason to believe that this has been the direct cause of the localization of many of the ore-bodies of the district. As the inclination of the ground-water surface is not so great as that of the strata, there would be raised at once at the point where the ground-water level intersected the basal plane of the Coal Measures a formidable barrier that would impose extensive impounding conditions. The Joplin district appears to be such an area.

In the central Missouri mining-district the series of parallel synclines gives conditions like those of Joplin, with the Coal Measures margin not far away. As already stated, the ore-

bodies appear to be confined entirely to the bottoms of the structural troughs. With this conception, systematic prospecting should make this a much more important mining-district than it is at present.

In the southeastern district the general conditions are also like they are in the southwestern area. Corresponding to the warped surface is the uneven erosion-plane of unconformity at the base of the Cambrian rocks. Instead of porous zones caused by brecciated cherts and limestones are the porous dolomites. The ores are "disseminated" through particular beds in place of formed into "veins" along underground channels filled with débris. The profile representing the surface of the Grand Falls chert, given in Fig. 18, would answer, with no special modification, for the generalized cross-section of the southeastern lead-field. On the surface in the southeast would be an unconformity-plane instead of a warped stratigraphic plane.

XVII. Migration of Ores.

The most noteworthy feature regarding the geographic distribution of the ore-deposits of the Ozark region is, as already stated, that they are mainly confined to a relatively narrow belt which completely encircles the great dome at its very base. While this belt reaches practically unbroken around the entire dome, the unequal richness of the deposits, owing to local differences in geologic structure, has given rise to the general impression that the ore-deposits are accidentally located in four or five comparatively small and rather widely separated areas. This impression is further emphasized by the fact that, in consequence of the mining-camps being segregated, they have been regarded as distinct districts. The possible change of position of the ore-belt from time to time is a theme which is of great scientific interest as well as of no small practical significance. The geologic history of the Ozark region includes an episode, just before the last upraising into the present dome, during which the Coal Measures extending nearly or quite over the center of the area were beveled off. As the present dome arose, this thin layer of soft shales was rapidly peeled off the hard sloping limestones beneath. A low Coal Measures escarpment, if it might be so called, everywhere faced the center of the dome, as it still does to-day.

Considering the Joplin ore-field alone, it was probably at one time located near the crest of the dome at Cedar gap, but as the Coal Measures were rapidly stripped off through general surface-erosion, it followed down, in the pitching syncline, the ever-retreating margin of the shale-beds that continually acted as a barrier, or dam, to impound the vadose waters, or to raise the local ground-water level. Doubtless many minor ore-bodies now existing between Cedar gap and Joplin once belonged to the main ore-field, but, being cut off from the parent body in some way or other, remained intact, without being notably weathered or carried away in solution to other and lower levels, or out of the district altogether. Thus, as the Coal Measures margin retreated down the slope the ore-belt also continually migrated in the same direction, until its present position was reached.

XVIII. Recapitulation.

The salient features of the foregoing discussion may be briefly summed up. It appears that:

1. The Ozark ores are mainly strictly vadose deposits.

2. The geographic distribution of the main ore-deposits is circumscribed, the belt in which they are confined forming a continuous circle around the base of the great dome.

3. The ore-deposits have an intimate relationship to definitely determined geologic structures.

4. The general form of geologic structure, irrespective of origin, is the basin, or syncline, which may be formed in a number of very different ways—flexing, warping, faulting, or erosion.

5. The genetic relationship of ore-runs to buried relief-features at the base of the Coal Measures does not obtain.

6. The vertical range of workable ore-bodies may be expected to be very limited in depth, nowhere exceeding about 300 ft.

7. The frequent oscillation of the local ground-water level is an important factor in all mining-operations of the region.

8. The direct relation of the ore-bodies to fault-lines is not, by any means, a proved fact, or, at best, is of rare occurrence.

9. The primary source of the ore-materials is a factor the importance of which has been very greatly overestimated, and is of no significance in practical mining-operations.

10. The ore-zone practically corresponds to the normal zone of enrichment, but there is in the Ozark region really no notable oxidized zone and no deep zone of low-grade ores.

11. The present ore-bodies, as regards their geologic age, are very recent accumulations.

12. The controlling factor, above all others, in the localization of the ore-bodies is the basin-structure, causing a local impounding of the subterranean waters with their metallic salts in solution, and the determination of the boundaries of this structure is the all-important point to be kept in mind in systematic prospecting and in the planning of all mining-operations throughout the region.

A Graphic Solution of Kutter's Formula.

BY L. I. HEWES AND JOSEPH W. ROE, NEW HAVEN, CONN.

(New Haven Meeting, February, 1909.)

A GRAPHIC solution of Kutter's formula for the flow of water has been worked out by Dr. L. I. Hewes in connection with his course in Graphic Computations, given in the Sheffield Scientific School, Yale University, which may be of interest to those who deal with canals, flumes and streams.

Kutter's formula, $$V = \left[\frac{41.66 + \frac{1.811}{n} + \frac{0.00281}{S}}{1 + \left(41.66 + \frac{0.00281}{S}\right)\frac{n}{\sqrt{R}}} \right] \sqrt{RS},$$

is probably the most accurate of those proposed determining the flow of water. But it is extremely cumbersome, and if many calculations are to be made their solution becomes drudgery. The purpose of this chart, Fig. 1, is to furnish a rapid solution, sufficiently accurate to be well within the limits of the observed data.

The chart consists essentially of three parallel scales, one for the mean velocity, V, one for the hydraulic radius, R, and one for the slope; a diagonal line, not graduated; and a set of curves for the various values of n, the factor which involves the na-

ture and condition of the bottom. Usually, R and S have been determined, n assumed, and it is desired to calculate V. In this case the diagram is read as follows: From the given value of R on the middle scale project horizontally to the curve of the n-value assumed; thence project vertically to the diagonal. A straight-edge through the point on the diagonal so found and the point on the S-scale determined by the given conditions will cut the V-scale at the required mean velocity. The chart furnishes a means of determining any one of the four factors, the other three being known. This would be useful where it is desired to find a coefficient, n, corresponding to a given set of conditions, as the solution of Kutter's formula for this factor is especially troublesome.

The range of values covers all but the most extraordinary conditions. The graduations on the V-scale range from 0 to 25, far beyond any velocity encountered in practice; the scale of hydraulic radii ranges from 0 to 50 ft.; the slopes range from 1 ft. in 16 ft. to 1 ft. in two miles; and the n-curves cover all the values given in any of the books of reference.

The accuracy of the chart is about uniform for the various conditions encountered. Where the n-curves are acute to the R-scale the scale itself is very coarse, thus compensating for any error due to the obliquity of the curves. Opposite the upper end of the R-scale, where its graduations are finer, the n-curves have turned and are nearly parallel to it, so that an error in reading the R-scale has but little effect on the location of the pivot-point on the diagonal. Each of the curves has been checked by calculations, and the mean error of the total readings was only 0.53 per cent. The V-scale, on which the results are usually sought, with reasonable care in reading, will give results correct to within two or three units in the third significant figure, an accuracy well within that of the original data. The selection of n, for instance, is an element which introduces an uncertainty largely in excess of this.

Other graphic solutions of this formula have been worked out, but they nearly all involve a set of diagrams, one for each value of n. So far as we know, there have been none where the entire range of all factors has been included as conveniently in one diagram.

The Hammond Mining and Metallurgical Laboratory of the Sheffield Scientific School, Yale University.

BY LOUIS D. HUNTOON,* NEW HAVEN, CONN.

(New Haven Meeting, February, 1909.)

THE Hammond Mining and Metallurgical Laboratory is the gift of Prof. John Hays Hammond to the Sheffield Scientific School of Yale University. Professor Hammond was graduated from this school in the class of 1876, and has always shown great interest in its welfare and progress. In 1903, he offered to build and equip for it a mining and metallurgical laboratory. This offer was gratefully accepted by the trustees; and in October of the same year the architect, W. Gedney Beatty, and I were employed to draw up plans for the building, which is the largest of its kind in this country.

The Treasurer's report shows the total gift of Professor Hammond, including land, building, and equipment, to have been $128,741.53. This does not include $5,000 given by Professor Hammond to defray incidental expenses. The detailed cost of the machinery and installation, included in the above sum, is as follows:

Machinery,		$9,846.00
Pipe and fittings,		925.05
Hangers, shafting, pulleys, belts, etc.,		357.83
Steel-work, tanks, launders, etc.,		272.19
Screens for sizing ore,		411.37
Lumber,		1,896.36
General Supplies:		
Hardware,	$135.96	
Tools,	263.66	
Paints,	204.63	
Electric and incidental,	439.57	
		1,043.82
Machine-shop material,	259.44	
Labor,	722.29	
		981.73

* Professor of Mining and Metallurgy, Sheffield Scientific School, Yale University.

Foundry-work, castings, etc.,	$346.84
Freight, carting, and teaming,	397.38
Foundations, labor, and material,	1,257.37
Labor, construction, and installation,	9,355.09
General expense,	314.96
Total	$27,405.99

Requirements of the Laboratory.

Before planning the building, the mining-laboratories and ore-testing plants in the East were visited, and blue-prints of the Western and Canadian mining-laboratories were studied carefully. The methods of instruction and laboratory-work required of the students at different mining-schools were also carefully compared. After the completion of this preliminary work it was found that the equipment and laboratory-work required were different in the several universities, and that the design and equipment of the Hammond Laboratory would depend entirely upon the requirements placed upon it. A consultation with Professor Hammond resulted in the decision to erect a building to fulfill four requirements:

1. Teaching of the theory and practice of assaying, metallurgy, mining, and ore-dressing.

2. Testing of ores on a small scale to determine the most economical method of treatment.

3. Handling of ores on a commercial scale in order to verify the results obtained from the preliminary testing.

4. Research-work and investigations by the instructors or by professional men to develop new processes or perfect new machines.

The first requirement called for lecture-rooms and lecture-tables fitted with gas, water, and electric power for class-room experiments; also a museum for mining- and metallurgical models and the products of mills and smelters.

The second requirement, space for small ore-testing machinery.

The third requirement, space for full-sized machines, and floor-space for the installation of new machinery without changing the proposed arrangement.

The fourth requirement, and the most important for the advancement of the profession, space equipped with power, gas, and water for investigations on a large or small scale.

Erection of the Building.

The building was designed with the above objects in view, and the plans were accepted in May, 1904. The building-contract called for the building, and the furnishings for the library, lecture- and research-rooms, and also the desk-rooms for assaying and ore-testing. Ground was broken in June, 1904, and the building was ready for occupancy in November, 1905. During the autumn and winter of 1905 the assay-furnaces and small ore-testing machines were installed, and the supplies for the department of assaying were purchased. The first course in assaying and ore-testing was given the following spring. While this work was in progress, museum-cases and the balance of the furnishings were designed and built, and plans were drawn for the arrangement of the ore-dressing machinery, the order for which was placed on July 1, 1906, with the Allis-Chalmers Co. The first delivery of machinery was made in October, 1906, and in January, 1907, the first rock was broken in the crushing-room. In June the mill-wright, A. A. Watson, completed his work, and the first ore was milled during that month.

The building has a frontage of 84 ft. on Mansfield Street, and a depth of about 200 ft., extending back to the Northampton Division of the N. Y., N. H. & H. R. R. The detailed dimensions of the various rooms can be derived from the plan, Fig. 1. The lot upon which it is placed has a frontage of 100 ft. The railroad company built a siding at the rear of the building for the delivery of ores.

The building is constructed of fire-proof materials throughout, consisting of granite, limestone, brick, concrete, and steel. All exterior walls are lined with hollow brick, which was coated, before plastering, with anti-hydrene. The ground floor of the entire building is of cement, 5 in. thick. All steam-, gas-, water-, and electric conduits are exposed. The roof of the building, consisting of terra-cotta book-tile covered with slate, is supported by heavy iron trusses.

Description of the Building.

Floor-plans and sections of the Hammond Mining and Metallurgical Laboratory are given in Fig. 1, which shows also the arrangement of the different pieces of apparatus.

The ground floor of the building is divided into three por-

tions. The front portion consists of a basement and two floors. The floors, built of steel, hollow brick, and concrete, are divided into various rooms by partition-walls of hollow brick. The stairs connecting the floors are of steel with slate steps.

The second or laboratory portion of the building, adjoining the front portion, was designed to furnish as much light and air as possible, allowing for an increased floor-area. The side-walls are 27 ft. high, and the roof in the center of the building is 44 ft. high. In the center of the roof, for ventilation, there are monitors with sash windows on each side, which are opened and closed by gearing and chains extending to the floor. The windows in the side-walls are 5 ft. apart and are 20 ft. high by 5 ft. wide. These windows are divided in the center for the building of mezzanine floors along the outside walls as soon as the increased floor-area is required. Around the windows are steam-coils for heating, and from the roof-trusses are suspended arc-lamps for lighting. Between the windows and every few feet along the transverse walls are incandescent-lamp outlets, to which lamps can be attached to provide light for inspecting the machines.

This portion of the building is subdivided by two transverse walls into three rooms, having a total floor-area of about 12,000 sq. ft., which can be increased 10,000 sq. ft. by building the mezzanine floors planned. Eight chimneys were built in the transverse walls, two of which are lined with fire-brick. The floor of the large room slopes to a central sump, 12 ft. square by 3 ft. deep, with 12-in concrete sides and bottom. The floor of the sump slopes to a depression in one corner, which has a connection with the sewer, controlled by a gate-valve. In order to prevent the seepage of water into the sand under the building, the sump was lined with several layers of tarred paper and felt, held together with hot tar and covered with cement. The sump is connected with the main water-supply, and a pump is provided to enable the water to be used to supply the tank on the top floor. The floors in the other rooms are horizontal, with the exception of the space devoted to ore-testing, which slopes to open drains. In buildings of this character it would be better to have all the concrete laboratory-floors slope to an open drain for convenience in washing. Some of the concrete floors have been painted and others oiled to prevent dust from the surface-

wear of the concrete. Both methods have proved satisfactory. The paint gives a much better appearance but requires repainting, whereas the oil shows no wear.

The third portion of the building is triangular, conforming to the shape of the lot. It is divided into an upper and a lower floor: the former, a crushing-room; the latter, a boiler-room for steam-heating and a repair-shop.

Library, Museum, Etc.—The front portion of the building is devoted to the library, museum, lecture-rooms, and research-rooms. The first floor contains the library, museum, and office.

The library, to the left of the hall, has two windows facing east and two facing south. The room is furnished with shelves and two large tables, lighted by individual electric lamps and a ceiling-cluster.

The museum, across the hall from the library, has six windows. This room contains museum-cases and chests of drawers for student collections. It is intended to place in the museum typical flow-sheets of mills, and in the cases and drawers the various products from these mills. This arrangement will assist the student in studying the practical handling of ores from various districts. Here also will be placed models and photographs of mining-, metallurgical, and milling-plants and machinery.

The office is connected with the library and the hall.

Between the office and the museum and directly in front of the entrance are a hall-closet and an office-closet. Ultimately these closets will be used as a hallway connecting the entrance-hall with the mezzanine floors to be built later in the second or laboratory portion of the building as soon as this extra space is required.

On the second floor are two lecture-rooms, an instructor's room, and a hall-closet. Both lecture-rooms are furnished with chairs on stepped floors, lecture-tables fitted with gas-, water-, and electric-light connections, and an electric switch for power. The rooms are ventilated through a grating in the ceiling, which connects with a sky-light in the hall-closet.

The large lecture-room can accommodate 112 students. The lecture-table, 18 ft. long, contains large flat drawers for charts and maps. Back of the lecture-table are eight museum-cases with closets below. The four central cases have sashes

of slate for blackboard-work; the four outside cases have sashes of glass. There are also two connections for a lantern, one at the side of the lecture-table, the other in the rear of the room.

The small lecture-room can accommodate 45 students.

The instructor's room at present contains an electric blue-print frame and drawing-tables.

The basement, on the ground floor of the building, is subdivided into research-rooms, supply-rooms, locker-rooms, shower-baths, and lavatories, all heated by steam-coils suspended from the ceiling, and lighted by ceiling-clusters.

The two corner rooms are devoted to research-work. These rooms are furnished with desks, hoods, and electric power-plugs for power or electric-furnace work. Each room is lighted by three windows, individual electric lights on the desks, ceiling-clusters, and extra light-outlets in the walls for special work. The desks are equipped with water, gas, and electricity.

Adjacent to the southeast research-room is an instructor's room, equipped with desks for use in assaying. This room connects with the central hall and the assay-laboratory. The students' samples for assaying are stored in this room, together with the assay-records.

Adjacent to the northeast research-room is the balance-room, connecting with the assay-laboratory. This room is equipped with nine Becker button-balances, two Keller button-balances, and one Becker analytical balance. Light is furnished by one window, and each balance has an individual electric light.

The supply-rooms and dark-rooms are interior rooms connecting with the central hall. The supply-rooms contain shelves and drawers. The dark-room is furnished with a slate desk, water, gas, electric lights, and an exhaust-fan for ventilation.

The students' locker-room, for changing clothes, is in the front of the building, connecting with the central hall. This room can accommodate 80 lockers, 56 of which have been already built. Connected with the locker-room are two shower-baths and a lavatory.

The instructors' lavatory, containing lockers and a shower, connects with the hall, and is adjacent to the southeast re-

search-room. Hot water is supplied to the showers from a Rudd automatic gas water-heater.

Ore-Testing Laboratory.—The first laboratory-room, entered from the basement hall, is devoted to ore-testing and assaying. On the south side of the room, the equipment for ore-testing consists of eight working-tables covered with linoleum, and furnished with open drains to the floor. Above the tables are water-pipes furnished with dial plug-cocks leading from a reservoir on the wall, whereby a constant pressure is maintained. Adjacent to the windows are one gas drier and two steam driers. The apparatus installed consists of five Vezin jigs, five glass classifiers, three Munroe tables, sizing-screens, gold-washing and amalgamating-pans, *bateas*, and accessories necessary for the testing of ores.

Assay- and Metallurgical Laboratory.—On the north side of the first laboratory-room, 18 assay-desks were installed when the building was completed, and last summer 24 more, making a total of 42. The original desks were covered with slate and contained lockers, drawers, shelves, and desk-lockers for pulp-balances. The desks installed last summer differ by the addition of a cupel-drawer and the omission of the balance-locker. This arrangement reduces the number of balances required by one-half, and also gives the student a tray for cupels which otherwise would have to be kept in the locker below. The desks are placed in six sections; the three new sections contain eight desks each, and the three original ones contain six desks each, with three deep slate sinks at the ends. The desks have individual electric lights, are supplied with gas, and are subdivided to accommodate 84 students in two divisions. Distilled water is supplied from bottles supported on the top shelves. To prevent the stoppage of the sink-drains by ores from the preliminary vanning, and to prevent the destruction of the waste-pipes by acids, a 2-in. overflow is placed in each discharge-pipe, and the bottom of the sink is covered with broken limestone. The balance of the equipment in the desk-room consists of a hood, pulp-balances with individual sets of weights, and the general supplies required by an assayer. The students are charged, at cost, with a complete outfit for assaying, including weights, crucibles, etc., and at the end of the term the supplies not used and in good condition are returnable and full credit is

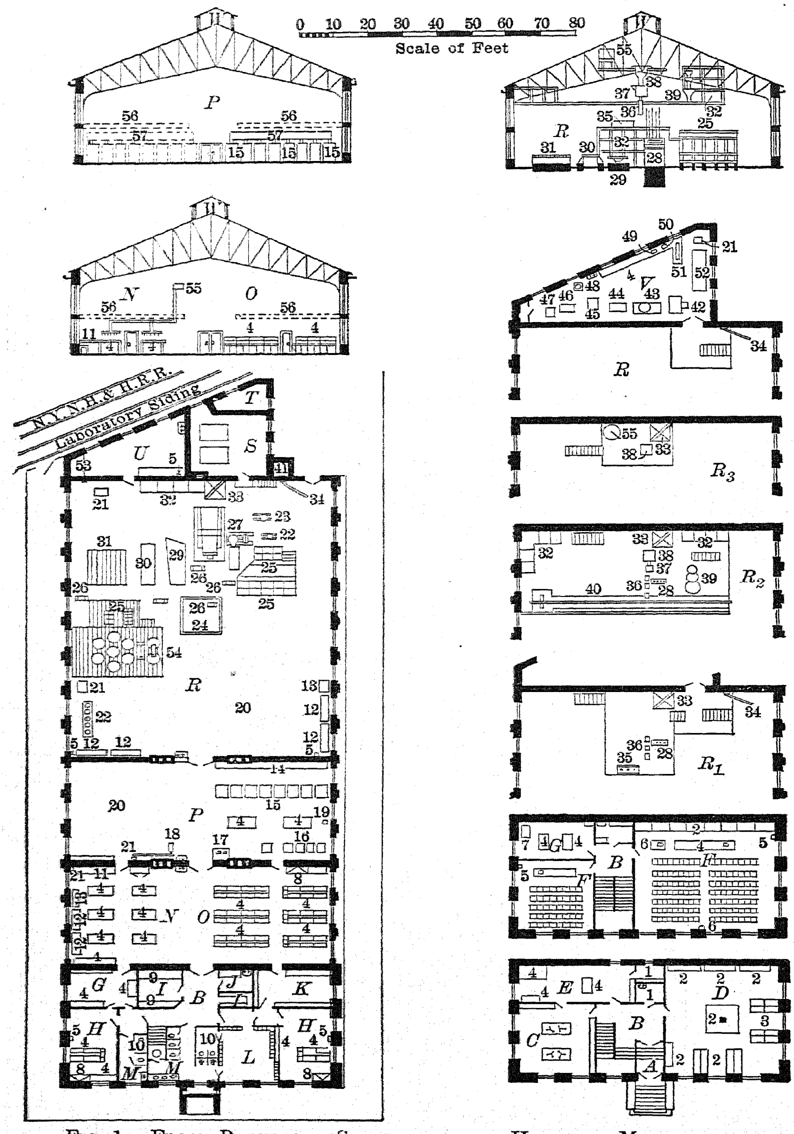


FIG. 1.—FLOOR-PLANS AND SECTIONS OF THE HAMMOND MINING AND METALLURGICAL LABORATORY.

Rooms and Apparatus of the Hammond Mining and Metallurgical Laboratory.

Rooms.

A. Entrance.
B. Hall.
C. Library.
D. Museum.
E. Office.
F. Lecture-rooms.
G. Instructor's-rooms.
H. Research-rooms.
I Supply-rooms.
J. Dark-room.
K. Balance-room.
L. Locker-room.
M. Lavatories.
N. Ore-testing room.
O. Assay-desk room.
P. Furnace-room.
R. Milling-room, ground floor.
R_1. Milling-room, first floor.
R_2. Milling-room, second floor.
R_3. Milling-room, third floor.
S. Boiler-room.
T. Coal-storage.
U. Repair-shop.
V. Crushing-room.
W. Monitors.

Apparatus.

1. Closets.
2. Glass cases.
3. Drawer-cases.
4. Desks.
5. Electric-power switches.
6. Electric lantern.
7. Electric blue-print frame.
8. Chemical hoods.
9. Shelves.
10. Showers.
11. Five Vezin jigs.
12. Steam driers.
13. Gas driers.
14. Coal-bins.
15. Soft-coal furnaces.
16. Gas-furnaces.
17. Crucible-furnaces.
18. Roots blower.
19. Bullion-rolls.
20. Space to grow.
21. Motors.
22. Amalgamating-pans.
23. Clean-up pan.
24. Sump.
25. Harz jig.
26. Centrifugal pumps.
27. Huntington mill.
28. Five-stamp battery.
29. Wilfley concentrator.
30. Frue vanner.
31. Wilfley slimer.
32. Ore-storage bins.
33. Platform-elevator.
34. Crane.
35. Hydraulic classifiers.
36. Trommels.
37. Vezin sampler.
38. Challenge feeders.
39. De-watering-cones.
40. Main line-shafting.
41. Ash-pit.
42. Blake breaker.
43. Gyratory breaker.
44. Dodge breaker.
45. Rolls.
46. Abbé pebble-mill.
47. Krupp ball-mill.
48. Sample-grinder.
49. Case breaker.
50. Abbé jar-mill.
51. Power bucking-board.
52. Shaking-screen.
53. Main switch-board.
54. Cyanide-plant.
55. Water-tank.
56. Proposed mezzanine floors.
57. Furnace-flues.
58. Proposed furnaces.
59. Laboratory R. R. switch.

given. The laboratory fee, which covers reagents and coal, is $15 for the course.

The second laboratory, adjacent to the first on the west, known as the furnace-room, has an available floor-area of 2,300 sq. ft., with a passageway of 10 ft. in the middle. As originally planned, this room was to have been devoted to assaying and metallurgy, but it has been found that the entire space will have to be devoted to assaying. The furnaces for metallurgical work, when built, will be installed on the proposed mezzanine floor, which will likewise have an area of 2,300 sq. ft. The two west windows in this room are door-windows, through which the soft coal is delivered to the steel coal-bin for the assay-furnaces. The bin is 3 ft. wide, 30 ft. long and 5 ft. high. Eight soft-coal, double-muffle assay-furnaces are installed 5 ft. in front of the coal-bin. Built of brick and lined with standard linings purchased from the Denver Fire Clay Co., these furnaces are similar to those used in the large mining- and metallurgical works in the West. Each furnace has its equipment of tools for firing and muffle-work. The students fire their own furnaces, and are responsible for the furnaces assigned to them. Facing the soft-coal furnaces are five muffle-furnaces, built by the American Gas Furnace Co., and two crucible coke-furnaces. The gas-furnaces were purchased through, and erected free of charge by, the New Haven Gas Co. The crucible-furnaces were designed and built in the laboratory. Air is supplied to the gas-furnaces by a Roots rotary blower operated by a variable-speed motor. This blower is of sufficient size to furnish air to 12 gas assay-furnaces or to a small cupola-furnace. The balance of the equipment consists of tables, pouring-molds, anvils, and the tools necessary for furnace-work. The floor-area in this room can accommodate 27 soft-coal, 8 gas-, and 2 crucible-furnaces. If gas-, gasoline-, oil- or crucible-furnaces are used in preference to the soft-coal furnaces, a much larger number can be installed. Soft-coal furnaces are preferable for student-work, since they are used at large mines and metallurgical works. When the equipment is increased, a few gasoline-furnaces will probably be added in order that the students may become familiar with the operation of this type of furnace.

Milling-Laboratory.—The large room in the rear of the build-

ing, adjacent to the furnace-room, contains the concentrating-machinery. It has a central sump with the entire floor sloping to it. The concentrating-machinery is situated between the sump and the west wall. The cyanide-plant and five small amalgamating-pans are adjacent to the south wall. The remaining floor-area is reserved for new machines, dry-concentrators, and magnetic separators. To secure the desired fall for the ore, it was necessary to erect three floors adjacent to the west wall. These floors will not interfere with the mezzanine floors to be built later on both sides of the room. The concentrating-machines were placed on separate concrete foundations extending 2 ft. below the ground floor and from 6 to 12 in. above the floor, depending on the machine. The stamp-mill foundation is a block of concrete, 8 by 14 ft. and 6 ft. deep, with a mortar-block of concrete, 5.5 ft. high. The machines are so arranged as to make their sequence interchangeable. The concentrating-machines on the main floor can be connected with centrifugal pumps, allowing the products to be sent to any other machine or to de-watering-cones placed on the top floor, the products of which, in turn, can be fed to classifiers or to any part of the room. The machinery can be operated for the testing of ores in car-load lots, or a small amount of ore can be circulated over one machine, allowing the student to study its adjustments.

The arrangement of the machinery installed is as follows: In the center of the room adjacent to the west wall is a platform-elevator for elevating ores to the various floors and the crushing-room. The five-stamp battery, with stamps each of 500 lb. and amalgamating-plates, is placed directly in front of the elevator. The battery can be fed either by a Challenge feeder or by hand. The pulp from the plates discharges to an amalgam-trap, and can then be fed to a centrifugal pump discharging to any machine or to de-watering-pans, the water being returned to the sump. To the left of the stamp-mill are the concentrating-tables, consisting of a Wilfley table and a slimer, both donated by the Mine & Smelter Co., and a Frue vanner. The products from the tables can be fed to one of two centrifugal pumps placed adjacent to the tables, or to de-watering-pans. To the right of the stamp-mill are placed a Huntington mill, jigs, an amalgamating-pan, and a clean-up pan. The Huntington mill is 3.5 ft. in diameter, with an apron-plate for amalgamation. The pulp from

the Huntington mill can be fed to a fourth centrifugal pump or to de-watering-pans. The jigs, of the Harz type, one of three and one of five compartments, 18 by 24 in., were designed and constructed in the building. They are elevated above the floor-level, allowing the products to be drawn off into de-watering-pans or to go directly to a centrifugal pump. The amalgamating-pan and clean-up pan are placed directly back of the jigs.

The first floor, at an elevation of 12 ft. above the ground floor, contains three Brown hydrometric classifiers. This floor is also the feeding-floor to the Challenge feeder preceding the stamp-mill. The second floor, at an elevation of 20 ft., contains three de-watering-cones. Between the second and third floors are a Challenge feeder, a Vezin sampler and three sets of trommels. The water-tank supplying the mill is on the top floor, and receives its supply of water from either the sump or the city supply.

The cyanide-plant consists of two solution-tanks, three ore-tanks, two 3-compartment precipitating-boxes, two sump-tanks, and a centrifugal pump connecting the sump-tanks with the solution-tanks. One of the ore-tanks is equipped with mechanical stirrers, and a second tank can be connected with a centrifugal pump for agitation. A Johnson filter-press, donated to the laboratory by the manufacturers, the John Johnson Co., is adjacent to the cyanide-plant. On the platform containing the precipitating-boxes are two Harz jigs, one of three and one of five compartments 9 by 9 in. Adjacent to the cyanide-plant are five small amalgamating-pans for the testing of ores. The remainder of the equipment consists of motors, four large steam driers, one gas drier, and ore-bins. Six centrifugal pumps are used: four for elevating ores, one with the cyanide-plant, and one to supply the water-tank. Water- and gas-pipes, fitted with plugs every few feet, extend along both outside walls. In three corners of the room are electric switches for power. The concentrating-machinery is driven from a main line of shafting belted to a 35-h.p. motor. The cyanide-plant and amalgamating-pans are operated in like manner by a 5-h.p. motor. The ore-bins are adjacent to the west wall back of the concentrating-machinery. It is intended to store car-loads of ore on the narrow strip of land south of the building.

Crushing-Room, Boiler-Room, Etc.—The crushing-room has a floor-area of 950 sq. ft., which may be extended to the north,

giving a total area of 1,500 sq. ft. The heavy machinery along the interior wall consists of a Blake breaker, 10 by 7 in., tight-and-loose pulleys, driven from main line of shafting; a Gates gyratory breaker, size No. 0, style D, friction-clutch on breaker, driven from main line of shafting; a Dodge breaker, 4 by 6 in., tight-and-loose pulleys, driven from main line of shafting; crushing-rolls, 12 by 12 in., belt-driven from short shaft connected to main line of shafting by friction-clutch; an Abbé pebble-mill, 30 by 19 in., tight-and-loose pulleys, driven from countershaft; a Krupp ball-mill, tight-and-loose pulleys, driven from countershaft. The two machines last mentioned were gifts of the Abbé Engineering Co. and Thos. Prosser & Sons. The machinery along the exterior walls consists of a sample-grinder, coffee-mill pattern, tight-and-loose pulleys, driven from countershaft; a Case laboratory-breaker, a jar-mill, and a power bucking-board, all driven from second countershaft; a shaking-screen, with an adjustable stroke, speed, and inclination, driven from a double-cone friction-pulley. The remainder of the equipment consists of various samplers, screens, bucking-boards, and supplies necessary for the breaking, crushing, and sampling of ores. The main line of shafting is driven by a 15-h.p. motor.

Below the crushing-room the floor-space is subdivided into a repair-shop, a boiler-room, and a coal-bin. The repair-shop, triangular in shape, contains the general switch-board supplying the building with electricity. An electric switch provides the current for operating a 5-h.p. motor in the northeast corner. The room is lighted by three windows and ceiling-clusters. It is intended to install in this room a lathe, drill-press, and other machines for repairs and construction as required.

The boiler-room contains two boilers for heating the building with steam. This room is lighted by three windows and electric lights. In the ceiling is an opening to permit the radiated heat from the boilers to enter the crushing-room.

In the northwest corner is the coal-storage, with windows on the railroad-siding and yard. It is of sufficient size to hold a car-load.

Floor-Areas.

The total floor-area in the building amounts to 22,871 sq. ft.; the front portion contains 6,691 sq. ft. and the laboratory por-

tion 16,180 sq. ft. By the building of the mezzanine floors, planned in the original design of the building, this floor-area can be increased by 10,750 sq. ft., making a total available area of 33,621 sq. feet.

The subdivisions of the area are as follows:

Front Portion.

Basement.	Sq. Ft.	*First Floor.*	Sq. Ft.
Research-room, H,	336	Library,	420
Research-room, H_1,	336	Museum,	1,188
Instructors,	216	Office,	396
Balance-room,	264	Closets,	100
Stock-room,	168	Total,	2,104
Sample-room,	55		
Dark-room,	77	*Second Floor.*	
Locker-room,	288		
Lavatory-room, M,	283	Lecture-room,	1,554
Lavatory-room, M_1,	120	Lecture-room,	506
Showers,	40	Instructor's-room,	264
		Closet,	80
Total,	2,183		
		Total,	2,404

Laboratories.

	Sq. Ft.	Sq. Ft.
Ore-testing,		1,320
Assaying:		
Desks,	1,320	
Furnaces,	2,640	
Balances,	264	
		4,224
Milling, ground floor,	6,400	
Floors,	2,600	
		9,000
Crushing-room,		950
Total,		15,494
Repair-shop, Furnace-room, etc.,		950

Available by Building Mezzanine Floors.

	Sq. Ft.	Sq. Ft.
Metallurgy:		
Furnaces,	2,300	
Desks,	1,150	
		3,450
Instructors,		1,150
Milling-room,		5,600
Crushing-room,		550
		10,750

Blast-Pressure at the Tuyeres and Inside the Furnace.

BY R. H. SWEETSER, COLUMBUS, OHIO.

(New Haven Meeting, February, 1909.)

AT the Buffalo meeting in October, 1898 (*Trans.*, xxviii., 865), our Secretary, Dr. Raymond, in speaking of the obstacles he had encountered in securing contributions to the *Transactions* from members in practice, said:

"It is the disinclination of many members to communicate incomplete or partial or preliminary results. They are going to write, 'some day,' a thorough and monumental paper; and meanwhile they prefer to communicate nothing. . . . It is the difficulty to which I have just referred that I have encountered in a wide correspondence with members, undertaken for the purpose of securing contributions of fact and opinion concerning the special subject of this discussion. . . . Almost none of them seem to recognize the value of preliminary, incomplete and indecisive reports of practice, as laying the foundation for a thorough inquiry."

This remark was made in immediate reference to a discussion concerning blast-furnace tuyeres; but, notwithstanding this appeal from the Secretary, little has been recorded regarding the distribution, the penetration, and the pressure of the blast at the tuyeres and inside the iron blast-furnace. I am therefore encouraged to contribute the following incomplete and crude observations of the blast-pressure inside the hearth.

At the suggestion of J. H. Frantz, the general manager of the Columbus Iron & Steel Co., a test was made at the East furnace of the company to determine whether there was any difference in blast-pressure at the different tuyeres. The apparatus used for this test was not sensitive enough to give accurate results. It consisted of a small pressure-gauge mounted on a $\frac{3}{8}$-in. gas-pipe. A hollow plug, fitting into the peep-hole of the tuyere-cap, prevented the blast from blowing-out during the test. The test-pipe was held in place by means of the handle on the outer end.

For the first series of tests the pipe was just long enough to extend through the blow-pipe to the nose of the tuyere. No checks or calculations were made to determine the effect on the gauge-readings of pressure produced by variations in the velocity of the blast.

To determine the pressure, one man lifted the tuyere-cap, and another pushed the pipe through the peep-hole till the hollow plug was tight in place, when a third took the reading on the gauge. It took but a few minutes to go all round the furnace. The first observations, made Jan. 24, 1908, were taken 30 min. after a very heavy slip in the furnace; and during the 19 min. required for the eight readings the blast-pressure in the bustle-pipe dropped 1 lb. The highest pressures were registered at tuyeres 2 and 3. All the tuyeres were 5.5 in. in diameter, except No. 8, which was only 4.5 in. There were obstructions in tuyeres 2 and 3, through which, consequently, there passed a smaller quantity of blast than through the others. The errors in these three readings, due to the velocity of the blast, would therefore be smaller. It is possible that all the pressures in the first test would be the same, if the proper corrections had been made; but, since no such corrections were made then or can be made now, it is not safe to draw conclusions as to the relative pressure at all the tuyeres. Although the end of the pipe became hot, it was not injured; and the same pipe served to test all eight tuyeres.

After the first experiment, it was decided to test the blast-pressure beyond the nose of the tuyere, and for this purpose longer $\frac{3}{8}$-in. pipes were provided, the first of which extended 7.75 in., and the others respectively 1 ft. 7.75 in., 2 ft. 6.5 in., 3 ft. 7.75 in., and 5 ft. beyond the nose of the tuyeres.

Jan. 28, 1908 (the furnace being hot and working well, except that tuyere 3 was plugged with iron and cinder, in consequence of a very heavy slip four days before), two tests were made with the end of pipe projecting 7.75 in. and 1 ft. 7.75 in. respectively beyond the nose of the tuyeres. There seemed to be little difference between the pressure at the nose of the tuyeres and the pressure a short distance inside the furnace. There was an apparent drop of 2 lb. in pressure between the bustle-pipe and the nose of the tuyere. The output of the furnace on that day was normal (250.4 tons).

Jan. 29, the pressure at the nose and at 2 ft. 6.5 in. beyond the nose was taken for all the eight tuyeres. In front of No. 2 the stock was a little "raw," and slag could be seen dropping down. When the pipe was pushed in beyond the nose of the tuyere the end became plugged with slag, and immediately the gauge showed 12 lb. pressure, due to the expansion of the air in the closed pipe. As soon as the cinder had been cleaned out of the pipe the experiments were continued.

Feb. 3, another test was made, with the end of the pipe projecting 3 ft. 7.75 in. beyond the nose of the tuyeres. The furnace was very hot and working well. The reading of the blast-gauge was taken as quickly as possible, but 3 ft. of the pipe burned off before it could be withdrawn. It was almost time for the first flush of cinder, and this circumstance, together with the high blast-temperature (860° F.), probably caused the burning-off of the pipe. At 1.47 p.m., just after the first flush, another set of tests was made. The blast-temperature was 830° F., and the pipe went into the furnace very easily and was quickly withdrawn. Three readings were taken before the end of the pipe had been melted off, and in each case the apparent pressure was higher inside the furnace than at the nose of the tuyeres.

The last test, Feb. 4, was made 20 min. after a cast. The pipe reached 5 ft. beyond the nose of tuyere 5. The furnace was very hot and working smoothly. The pipe was pushed into the blow-pipe until the end reached the nose of the tuyere, when the pressure was read from the gauge, and the pipe was quickly pushed into the furnace almost to the center of the hearth. There was just time to catch the reading of the blast-pressure before 5 ft. 1 in. of the pipe had melted off from the end. The two readings showed a drop of only 0.3 lb. between the nose of the tuyere and near the center of the furnace. On this day the furnace made 292.7 tons. Further experimenting had to be postponed on account of the blowing-out of the furnace on Feb. 5.

The East furnace, where these tests were made, was 75 ft. high by 18 ft. bosh-diameter and 12 ft. hearth-diameter, and had one 4.5- by 12-in. and seven 5.5- by 12-in. tuyeres. The product (of "basic" and "malleable" iron) averaged about 250 tons per day. The lining was old (the inwall dating from

April, 1903, the bosh having been once re-lined since that date) and was worn out in places. After blowing-out, it was found that the bustle-pipe was in some places almost wholly choked with flue-dust.

The details of the tests described above are given in Table I.

TABLE I.—*Experiments on Blast-Pressure Inside the Blast-Furnace.*

Date, 1908. Output.	Time.	Total Length of Pipe.	Position of Inside End.	Tuyere.	Pressure at Nose of Tuyere.	Pressure in Furnace.	Pressure in Bustle-pipe.	Pressure at Engines.	Temp. of Blast.	Remarks.
					Lb.	Lb	Lb	Lb.	°F.	
Jan. 24. 213.7 tons pig.	11.27 a.m.	6 ft. 9 in.	At nose.	1	8.8	....	10.0	....	840	Flushing cinder during first two readings. Furnace had made a heavy slip at 10.45 a.m. Pressure dropped from 15 lb. at 10.46 to 9 lb. at 11.20 a.m. because the steam dropped 35 lb. in as many minutes.
	11.29			2	9.1	....	10.0	....		
	11.31			3	9.1	....	10.0	....		
	11.15			4	8.5	....	11.0	....		
	11.17			5	8.6	....	....	....		
	11.20			6	8.9	....	....	....		
	11.22			7	8.5	....	....	....		
	11.24			8	9.0	....	10.1	....		
	11.34			4	8.2	....	10.0	....		
	3.20 p.m.	7 ft. 4.75 in.	7.75 in. inside.	1	9.25	8.0	10.75	11.1	730	The little gauge used at tuyere records 0.3 lb. less than bustle-pipe gauge.
	3.25			2	9.5	8.0	10.75	11.1		
	3.29			4	8.0	8.0	10.75	11.1		
	3.30			5	8.1	8.0	10.75	11.0		
	3.31			6	8.0	7.9	10.75	11.0		
	3.32			7	7.8	8.0	10.75	11.0		
	3.33			8	8.8	8.0	10.6	11.0		
Jan. 28. 250.4 tons pig.	4.12	8 ft. 4.75 in.	1 ft. 7.75 in.	1	8.2	8.2	10.3	10.7	690	Clear tuyere, crooked blow-pipe.
	4.14			2	8.0	8.2	10.2	10.7		Tuyere half-filled, crooked blow-pipe.
	4.16			4	8.1	8.1	10.2	10.6		Clear tuyere, straight pipe.
	4.17			5	8.2	8.2	10.2	10.6		Clear tuyere, straight pipe.
	4.18			6	8.6	8.1	10.2	10.6		Clear tuyere, straight pipe.
	4.19			7	8.1	8.2	10.2	10.6		Clear tuyere, crooked pipe.
	4.20			8	8.1	8.0	10.2	10.6		Clear tuyere, 4.5 in.
	4.25			2	8.7	8.4	10.7	11.0		
Jan. 29. 250.9 tons pig.	3.07	9 ft. 3.5 in.	2 ft. 6.5 in.	1	7.8	7.8	10.0	10.6	740	
	3.09			2	7.5	12.0*	10.0	10.6		*End of pipe plugged with cinder.
	3.20			3	8.0	7.6	10.0	10.6		Tuyere partly plugged.
	3.13			4	7.2	7.6	10.0	10.6		
	3.15			5	7.9	7.5	10.0	10.7		
	3.16			6	7.5	7.5	10.0	10.7		
	3.17			7	7.5	7.5	10.0	10.7		
	3.18			8	8.2	7.7	10.0	10.7		4.5-in. tuyere.
	3.21			2	8.0	8.0	10.0	10.7		
Feb. 3. 244.2 tons pig.	9.06 a.m.	10 ft. 4.75 in.	3 ft. 7.75 in.	4	6.4	6.5	10.1	11.1*	860	*Furnace very hot and working well, pipe burned off 3 ft. back.
	1.47 p.m.			5	7.5	8.0	10.5	11.8	830	Just through flushing, pipe went in easy.
	1.51			6	7.5	7.6	10.5	11.7		
	1.52			7	7.7	7.8	10.5	11.7		
Feb. 4 292.7 tons pig.	11.44	11 ft. 9 in.	5 ft. 0 in.	5	7.5	7.2	10.3	11.0	860	20 min. after cast, 5 ft. 1 in. of pipe burned off almost immediately.

When either of the two furnaces of the company is blown in again, it is intended to make further experiments along this line. The furnace-lining will then be new and free from obstructions, and all stoves and blast-connections will be clean and of known diameter. Possibly by that time some better device for getting the true pressure will be developed and used.

Under the conditions of the above tests, the results obtained are not regarded as possessing much value in themselves, but they point the way to further investigations which may afford us important guidance in the determination of proper diameters for tuyeres, blow-pipes, and blast-connections. I hope this fragment of imperfect research will bring out reports of other experiments, and also stimulate similar inquiries under other conditions.

POSTSCRIPT.

The foregoing paper was submitted in manuscript to a member of the Institute, qualified by long practice and professional authority to give advice on this subject, but unable at this time

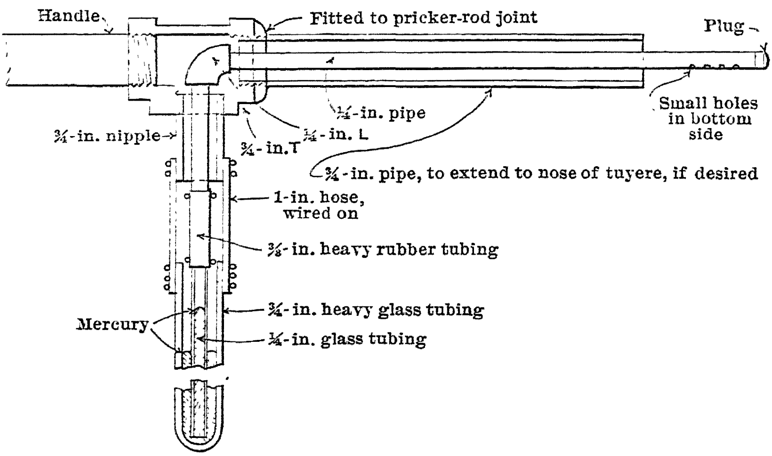


FIG. 1.—SKETCH OF ANNULAR U-GAUGE FOR DETERMINING DIFFERENCES IN PRESSURE.

to engage in a formal discussion of it, and therefore unwilling to offer a personal contribution to such a discussion. With his permission, I publish the following summary of his remarks:

I do not know of any other investigations along this line except some very thorough ones made by Fliegner on The Flow of Air Through Orifices, which have always been considered authoritative, and which take account of both the absolute pressure and the absolute temperature. J.E. Johnson, Jr., in his Notes on the Physical Action of the Blast-Furnace, gives an approximate formula[1] for the drop of pressure in passing through the tuyeres. For the pressure-drop through the tuyeres alone, I would trust this formula in preference to your experimental results, because the conditions of your experiment, as you frankly state them, were

[1] *Trans.*, xxxvi., 460 (1906).

so exceedingly bad, in comparison with what they could be made, and were made by Fliegner.

I think it is a mistake to take only the pressure in the bustle-pipe. What I should want to know in such a case would be the difference in pressure between the blow-pipe and the inside of the furnace, which can be easily determined by the use of a **U**-gauge, filled with mercury and arranged about as shown on the accompanying rough sketch, Fig. 1, the outer tube being filled with water above the mercury to above the top of the hose before screwing the nipple into the **T**. This would give direct readings of differences of pressure, which are very much more desirable than those of small differences in large pressures, taken from gauges which may or may not be accurate. The nicest form of **U**-gauge for this job is the annular one, in which one pressure takes effect in the central tube and the other in the annular space. This makes the difference extremely easy to measure, and compensates for any error in holding the gauge plumb, etc.

Another thing which occurs to me is, that, as a result of the high velocity of the entering blast, the suction effect on the open end of the tube will give an appreciable error, to avoid which, the test-tube should be plugged on the end and its side drilled full of small holes with rounded edges.

If you make any more tests of this kind, these two changes, in my opinion, would render your work much easier, and your results more accurate.

I suppose everybody will be at first inclined to reject, as due to errors of observation, the indication given by the experiments you report, that the pressure inside the furnace is higher than that at the tuyeres. But this is not necessarily true. The high velocity of the air-jet may be transformed back into pressure inside the furnace; in fact, that is what its velocity is for. This point also can be determined with the differential guage by using two concentric pipes, the outer one of which reaches to the nose of the tuyeres, and the other to any desired distance beyond that, in the furnace.

I have taken in the past a great deal of interest in this subject; aud I shall be glad to see your paper published, for the discussion that it ought to bring out. This would be all the more valuable if it should induce you, or some one else, to make more extended and accurate tests, in which the friction in the bustle-pipe and stocks was eliminated. The ordinary formula for the flow of air enables us to design bustle-pipes so that there need be no appreciable loss of pressure when they are clean; and, in a case of this kind, such an unnecessary loss only tends to becloud the value and diminish the interest of the investigation.

I beg to express my thanks to this friendly critic, whose suggestions possess the highest value for me; and I may be permitted to add that, even if my paper should receive no other discussion, this one result of its publication will, in my judgment, justify the wisdom of that invitation of our Secretary (quoted in its first paragraph) which encouraged me to write it.

The Coal-Fields of the United States.

BY MARIUS R. CAMPBELL AND EDWARD W. PARKER, WASHINGTON, D. C.

(New Haven Meeting, February, 1909.)

Description.

According to the estimates prepared by the U. S. Geological Survey, the area underlain by workable coal-beds in the United States is 496,776 sq. miles. Of this total area, 480 sq. miles contain the entire anthracite coal-fields of Pennsylvania. The bituminous coal-fields are estimated to be contained in areas aggregating 250,051 sq. miles. The grade of coal between bituminous and lignite, and which is designated by the Geological Survey as "sub-bituminous," is estimated to be contained within areas aggregating 97,636 sq. miles, while the areas containing lignite aggregate 148,609 sq. miles. The coal-fields are divided, for the sake of convenience in classification, into six main provinces, as follows:

1. *The Eastern Province*, containing the anthracite coal-fields of Pennsylvania and the bituminous coal-fields of the Appalachian region—namely, those of western Pennsylvania, Ohio, Virginia, West Virginia, Kentucky, Tennessee, Georgia, Alabama, and small outlying areas in North Carolina.

2. *The Interior Province*, containing the bituminous coal-producing regions of Michigan, Illinois, Indiana, western Kentucky, Iowa, Kansas, Missouri, Oklahoma, Arkansas, and Texas.

3. *The Gulf Province*, containing the lignite-areas of Alabama, Mississippi, Louisiana, Arkansas, and Texas.

4. *The Northern or Great Plains Province*, containing the lignite and sub-bituminous areas of North and South Dakota, eastern Montana, and northeastern Wyoming.

5. *The Rocky Mountain Province*, containing the bituminous and sub-bituminous areas of western Montana and western Wyoming, Colorado, Utah, and New Mexico.

6. *The Pacific Coast Province*, containing the areas of Washington, Oregon, and California.

During the last few years the Survey geologists have worked in all of these coal-areas, and have also been making careful estimates as to the quantity of coal contained in the beds when mining first began. In making these estimates care has been taken to ascertain how much of the supply is easily available, and how much is either not available under present mining- and market-conditions, or is available with extreme difficulty. According to these estimates, the quantity of coal contained within the known area of the United States, when mining first began, was 3,076,204,000,000 tons. Of this quantity a little less than two-thirds, or 1,922,979,000,000 tons, is considered as coal that is easily accessible or minable under present conditions, while slightly more than one-third, or 1,153,225,000,000 tons, is considered as non-minable under present conditions, or accessible with extreme difficulty. It should be remembered, however, that the quantity of coal given above as easily accessible includes the lignites and sub-bituminous coals of the Western States, of which approximately 530,000,000,000 tons, while easily accessible, cannot be considered available under present conditions, or those which may be anticipated in the near future. This reduces the original supply of easily accessible and available coal to approximately 1,392,979,000,000 tons.

The areas of the different provinces and the quantity of coal contained therein, when mining first began, are shown in Table I.

TABLE I.—*Tonnage (Short Tons) by Provinces and Accessibility.*

Province.	Area.	Original Coal-Supply.		
		Amount Easily Accessible.	Amount Accessible with Difficulty.	Total.
	Sq. Miles.	Tons.	Tons.	Tons.
1. Eastern	70,022	555,634,000,000	8,000,000,000	563,634,000,000
2. Interior	144,664	406,667,000,000	91,000,000,000	497,667,000,000
3. Gulf	84,300	13,045,000,000	10,045,000,000	23,090,000,000
4. Northern or Great Plains.	103,564	521,793,000,000	459,000,000,000	980,793,000,000
5. Rocky Mountain	92,396	414,740,000,000	574,280,000,000	989,020,000,000
6. Pacific Coast	1,830	11,100,000,000	10,900,000,000	22,000,000,000
Total	496,776	1,922,979,000,000	1,153,225,000,000	3,076,204,000,000

The distribution of this original supply of coal, according to grades and accessibility, is shown in Table II.

TABLE II.—*Tonnage (Short Tons) by Grades of Coal and Accessibility.*

Kind of Coal.	Area.	Original Coal-Supply.		
		Amount Easily Accessible.	Amount Accessible with Difficulty.	Easily Accessible and Available.
	Sq. Miles.	Tons.	Tons.	Tons.
Anthracite and bituminous....	250,531	1,176,727,000,000	505,730,000,000	1,176,727,000,000
Sub-bituminous	97,636	356,707,000,000	293,450,000,000	216,252,000,000
Lignite	148,609	389,545,000,000	354,045,000,000	
Total..........	496,776	1,922,979,000,000	1,153,225,000,000	1,392,979,000,000

The first mining of coal in a commercial way, in the United States, was in what is known as the Richmond Basin, a small area in the eastern part of Virginia. Small quantities of coal had been mined here in the latter part of the eighteenth century, and it was also in the latter part of the eighteenth and the beginning of the nineteenth centuries that efforts were being made to introduce anthracite coal for fuel-purposes. The first actual records of the production of Virginia coal were in 1822, in which year it was reported that 54,000 tons were mined. In 1820 (two years before) 365 long tons of anthracite coal, or one ton for each day of the year, had been shipped to distant markets. From these small beginnings of less than a century ago the production of coal has increased until in 1907 the total output of anthracite and bituminous coal approximated 500,000,000 tons. In 1837 the coal-production of the United States reached, for the first time, a total exceeding 1,000,000 tons, the output being reported from four States only, Pennsylvania, Virginia, Kentucky, and Illinois, although Maryland also was producing a small quantity of coal at that time. In 1840 the production amounted to a little more than 2,000,000 tons, the output being reported from 13 States. Ten years later, in 1850, the production amounted to 7,000,000 tons; in 1860 it was more than 14,000,000 tons; in 1870, more than 33,000,000 tons; in 1880, more than 70,000,000 tons; in 1890 it approximated 160,000,000 tons; in 1900 it was nearly 270,000,000 tons, and in 1907 it was 480,000,000 tons. The aggregate production to the close of 1907 has amounted to 6,865,097,567 short tons.

Up to the close of 1845 the total production of coal in the

United States was 27,700,000 short tons, and since that time the drain on the supply has practically doubled with each decade. The total production to 1845, and decennially since that time, was:

	Short Tons.
Up to 1845,	27,677,214
1846–1855,	83,417,827
1856–1865,	173,795,014
1866–1875,	419,425,104
1876–1885,	847,760,319
1886–1895,	1,586,098,641
1896–1905,	2,832,402,746
1906–1907,	894,520,702
Total,	6,865,097,567

It is estimated that for every ton of coal mined and sold half a ton is lost or wasted, so that the total production of 6,865,097,567 short tons to the close of 1907 represents an exhaustion of 10,200,000,000 tons, or 0.3 per cent. of the total original supply, or 0.7 per cent. of the coal which is easily accessible and available under present conditions. The total supply of easily accessible and now available coal left in the ground at the close of 1907 was 1,382,780,000,000 short tons.

Table III. shows the production of coal annually from 1846 to 1907; also, the average annual production by progressive decades for the same length of time, the latter having been prepared in order to eliminate minor variations due to abnormal conditions. The average annual increase in coal-production, figured from the average of progressive decades, is 7.36 per cent., and for the last five progressive decades—1894 to 1903 to 1898 to 1907—the rate of increase has been above the average.

Duration of Supply.

The total reserve of easily accessible and now available coal is estimated at 1,382,780,000,000 tons. The assumption that a constant output has been reached would be utterly unwarranted. On the other hand, the adoption of the flat rate of annual increase of 7.36 per cent. would involve the improbable assumption that the marvelous record of the past and present will be maintained in the future, and that the production will continue to approximately double every decade. Using the waste-allowance, on the basis of this constant rate of increase

TABLE III.—*Annual Production of Coal in the United States, by Single Years and by Average of Decades.*

Year.	Short Tons.	Average from	Short Tons.
1846	4,865,522	1846–1855	8,342,000
1847	5,286,067	1847–1856	9,210,000
1848	5,773,974	1848–1857	10,015,000
1849	6,448,831	1849–1858	10,835,000
1850	7,018,181	1850–1859	11,753,000
1851	8,734,525	1851–1860	12,512,000
1852	9,816,664	1852–1861	13,288,000
1853	10,570,288	1853–1862	14,055,000
1854	11,977,102	1854–1863	15,130,000
1855	12,926,673	1855–1864	16,293,000
1856	13,546,925	1856–1865	17,380,000
1857	13,340,189	1857–1866	18,925,000
1858	13,974,478	1858–1867	20,663,000
1859	15,633,175	1859–1868	22,552,000
1860	14,610,042	1860–1869	24,279,000
1861	16,488,012	1861–1870	26,122,000
1862	17,485,835	1862–1871	29,162,000
1863	21,319,062	1863–1872	32,558,000
1864	23,605,123	1864–1873	36,186,000
1865	23,792,173	1865–1874	39,086,000
1866	29,003,583	1866–1875	41,943,000
1867	30,724,422	1867–1876	44,371,000
1868	32,861,960	1868–1877	47,349,000
1869	32,904,360	1869–1878	49,857,000
1870	33,035,580	1870–1879	53,378,000
1871	46,885,080	1871–1880	57,222,000
1872	51,453,399	1872–1881	61,121,000
1873	57,602,480	1873–1882	66,331,000
1874	52,605,920	1874–1883	72,142,000
1875	52,348,320	1875–1884	78,897,000
1876	53,280,000	1876–1885	84,776,000
1877	60,501,760	1877–1886	90,816,000
1878	57,935,600	1878–1887	97,831,000
1879	68,105,799	1879–1888	106,903,000
1880	71,481,570	1880–1889	114,215,000
1881	85,881,030	1881–1890	122,844,000
1882	103,551,189	1882–1891	131,113,000
1883	115,707,525	1883–1892	138,691,000
1884	120,155,551	1884–1893	145,355,000
1885	111,160,295	1885–1894	150,413,000
1886	113,680,427	1886–1895	158,610,000
1887	130,650,511	1887–1896	166,441,000
1888	148,659,657	1888–1897	173,399,000
1889	141,229,513	1889–1898	180,531,000
1890	157,770,963	1890–1899	191,782,000
1891	168,566,669	1891–1900	202,973,000
1892	179,329,071	1892–1901	215,446,000
1893	182,352,774	1893–1902	227,672,000
1894	170,741,526	1894–1903	245,173,000
1895	193,117,530	1895–1904	263,281,000
1896	191,986,357	1896–1905	283,240,000
1897	200,229,199	1897–1906	305,457,000
1898	219,976,267	1898–1907	333,470,000
1899	253,741,192		
1900	269,684,027		
1901	293,299,816		
1902	301,590,439		
1903	357,356,416		
1904	351,816,398		
1905	392,722,635		
1906	414,157,278		
1907	480,363,424		

in production, the 1,382,780,000,000 tons available at the close of 1907 would be exhausted in 107 years, or by 2015 A.D. Against the use of the flat rate of increase it may well be contended that just as the rate of increase in population tends to diminish, so this rapid increase in per capita consumption of coal cannot persist, and a constant annual production will be reached. However, the figures set 50 years ago by statisticians for the probable constant annual production of coal in England have already been exceeded by more than 160 per cent.

Henry Gannett has made an estimate based upon a decreasing rate of increase calculated from 20-year averages of production. The use of 10-year averages is regarded as unsatisfactory, for the reason that one of the decades may consist mainly of a period of prosperity, while the preceding and succeeding decades contain periods of business depression. The 20-year period, however, is sufficiently long to include a period of prosperity with one of business depression. Taking the four 20-year periods since 1828, three rates of increase are obtained which show a rapid decrease. The hyperbolic curve, computed from these successive rates of increase, will indicate the constantly diminishing rate of increase for the successive 20-year periods. The result obtained by this method is that the easily accessible and available coal will be exhausted about the year 2027, and all coal by the middle of the next century. It is recognized that the data upon which this curve has been constructed are few, and that the curve is correspondingly weak. However, in the above estimate all the data have been given which it is possible to use, and this estimate is believed to represent the best use that can be made of the data at hand.

Inasmuch as the United States not only leads the world in present production of coal, but also apparently possesses the greatest reserve, and certainly is mining coal at much lower cost than any other country, the obvious tendency will be for European countries to look more and more to the United States for their coal-supply. Therefore, while our present coal-production and consumption are practically equivalent, the export of coal, unless prohibited by Federal legislation, must eventually become a factor and increase the coal-production in the United States beyond the demands of home consumption. On the other hand, powerful influences will come to bear upon

coal-production, which favor lengthening the life of the supply. Thus, it is to be hoped that more improved methods in the utilization of coal and increased efficiency per unit may act as a factor in reducing coal-consumption, and improved mining-methods should likewise decrease the waste percentage. The increased utilization of water-power should also tend to decrease coal-consumption. Again, as soon as the end appears in sight prices will rise and production diminish, and that progressively. This interference with the law of decreasing increase, produced by growing scarcity, will, of course, prolong the life of our coal-reserves, but at the same time will greatly hamper our industries dependent on this fuel.

With so many indeterminate factors whose importance is realized but cannot be measured, prophecy must possess a questionable value.

Waste in Coal-Mining.

The principal loss or waste attending coal-mining operations is that represented by the quantity of coal necessarily left in the ground as pillars to support the roof. In some cases it is also necessary to leave a foot or more of coal as a part of the roof, because of the unstable character of the material overlying the coal, which itself does not make a good roof. It has also been frequently the case that, where portions of the coal-beds have been of inferior quality, only the high-grade coal has been mined and the poorer material left. The coal left as pillars, or as portions of the roof, may be considered a necessary loss, but that which is left because of its inferior quality cannot be considered unavoidable waste in any sense, and is frequently of higher grade than coals mined and used in other portions of the country.

Enormous quantities of coal have been lost beyond recovery from the mining of beds lying below, the caving of which, upon the withdrawal of the pillars, has so broken the overlying strata as to render it impossible to recover the coal contained therein. This has been particularly the case in some of the coal-beds of western Pennsylvania, but much improvement has been observed in this regard within later years. Notwithstanding the improvement in this respect, it is probable that a large amount of coal will be wasted in the Western

States, where a great number of coal-beds are closely associated, and also where the intercalated strata are weak, forming poor roofs to the coal-mines.

There are no exact figures as to the actual loss or waste sustained through coal left in the mines in conducting the mining-operations, but it has been estimated that it amounts to 50 per cent. of the quantity produced and marketed. In some cases, through careful mining, and where the conditions are ideal for working, practically all of the contents of the coal-beds are recovered. In other cases, particularly when the beds are of enormous thickness, the recovery has not exceeded 30 per cent. of the contents. During the earlier days of mining in the anthracite-regions of Pennsylvania, it was estimated that only 40 per cent. of the coal was marketed. This was partly due to uneconomical methods of mining, and partly to the large amount of culm, for which there was at that time no market, and which was piled on the ground in unsightly mountains. At the time of the Anthracite Coal Waste Commission, which made its report in 1893, the maximum recovery was still considered to be 40 per cent. So far as underground workings are concerned, there has been no revolution in the methods employed since that time, but there has been a considerable improvement in the application of those methods, which has resulted in the recovery at the present time of a materially larger proportion of the coal in the ground than was the rule at that date. The earlier methods of mining consisted in leaving comparatively narrow pillars and in the mining of large rooms, the result being that the pillars were not strong enough to stand the pressure and they were crushed beyond recovery. It is now customary to use larger pillars between the rooms, which makes it possible to control the roof better during "robbing" operations, and to recover eventually a larger proportion of the contents of the bed.

Kentucky Fluorspar and Its Value to the Iron and Steel Industries.

BY F. JULIUS FOHS. LEXINGTON. KY.

(New Haven Meeting, February, 1909.)

CENTRALLY located with relation to the largest iron- and steel-producing districts of the United States, the fluorspar-deposits of Kentucky possess increasing interest and importance. As typical of the numerous fluorspar-mines of Kentucky and Illinois, I have selected for description the Memphis, because it and others of its immediate group exhibit the conditions characteristic of all.

I. THE MEMPHIS MINE.

This mine lies 5 miles NW. of Marion, in Crittenden county, Kentucky, a station on the Chicago and Nashville division of the Illinois Central R. R. For the first mile and a half out of Marion we see only Birdsville or Middle Chester sandstones of Mississippian age. Then we cross a fault of about 350 ft. displacement, and the Ste. Genevieve oölite sets in. Beyond the fault, oölite is the predominant rock, except that the hills to the north have a capping of Cypress or lowest Chester sandstone, 80 ft. below which is an outcrop of from 6 to 10 ft. of Ste. Genevieve (Rosiclare) sandstone. Occasional sink-holes are noticed, and as the mines are approached evidence of faulting is again seen. A line of crude head-frames and open-cut dumps, extending for more than a quarter of a mile, marks the Klondike vein. The open-cuts are from 15 to 20 ft. deep, and in most of them, from the surface down, was found the reddish gravel which is really fluorspar—some of it white or purple and crystal-clear (if the red clay is washed off), and some carrying iron and silica, and stained by waters from the decaying limestone walls. In one of these cuts, at a depth of 20 ft., white and purplish honey-combed fluorspar was found, in which the larger cavities are due to the leaching of limestone and calcite, and the minute ones to the leaching of zinc-blende.

The main shaft on the Klondike vein is 50 ft. deep. Descending this, we see Ste. Genevieve limestone walls at a depth of less than 25 ft., and between them beautiful snow-white fluorspar, instead of the red gravel found above between clay walls. At 35 ft. the vein is 6 ft. thick. It is vertical, and the strike is about N. 33° E.—the general trend of the veins of the Memphis group. The bedding-planes of the walls are horizontal, and, as there has been no displacement, these horizontal planes continue through the fluorspar; but near the center there remain some unreplaced fragments of limestone. The fluorspar at this point is granular, crystalline-massive, a little heavier than limestone, but not as heavy as zinc-blende. Its specific gravity is 3.3, and its hardness 4 (it will scratch calcite). Beautiful groups of semi-transparent white, and more rarely purple, cubic fluorspar crystals, sometimes interlocking and sometimes with a thin horizontal layer of clay separating the opposite groups, occur here. These crystalline aggregates found room in places along the bedding-planes existing at intervals in the limestone previous to its replacement by fluorspar. Where thinly laminated siliceous limestones, instead of massive pure limestones, such as those of the Memphis mine formation, are thus replaced, the horizontal layers of crystals are found attached to thin intervening layers of siliceous or jasperoidal fluorspar, forming a ribbon-structure. In diameter, most of the Klondike crystals measure 2 in. or less, and the largest about 11 inches.

At a depth of 50 ft. the vein narrows, and carries more calcite and a little pyrite. The calcite, generally associated with fluorspar, is usually milk-white, and shows an easy rhombohedral cleavage, while that of the fluorspar is octahedral or tetrahedral. Pyrite is rare in fluorspar-veins.

The Klondike vein can be traced NE. through many openings, in one of which it shows 2 ft. of fluorspar, zinc carbonate, and limestone. At another place, on higher ground, a tunnel exposes fluorspar-gravel with Rosiclare sandstone walls; and further on, where the land is more elevated, some prospect-pits show purple fluorspar cementing Cypress quartzose sandstone, or quartzite.

About 75 ft. SE. of the Klondike is a parallel vein carrying purple fluorspar, cementing Rosiclare sandstone at intersections. About 100 yd. NE., and higher up the hill, a generally

similar vein is found. There must be a fault at the base of the hill between this vein and the Klondike. Still further NW., where sink-holes abound, is another fault, and about 100 ft. beyond this is the Memphis vein, on which a 60° incline takes us to the 70-ft. level. The walls of the vein have about the same dip as the incline, and are of Ste. Genevieve limestone, the upper member of which, the Ohara, forms the SE. wall at the surface, whereas the opposite wall starts with the lower member of this formation, the Fredonia oölite. On the SE., at 70.5 ft. depth, the base of the Rosiclare sandstone, the middle member, is reached, showing the displacement to be 70.5 ft. The walls are grooved, and the hanging-wall is slickensided by the faulting.

At the 70-ft. level, the vein is very narrow for the first 12 ft. SE., but then suddenly swells. In a stope 14 ft. above the level we find 8 ft. of sheeted fluorspar, with more or less galena between the sheets, and clay in the seams; and 20 ft. further on the stopes show about 14 ft. of fluorspar, irregularly blocked by thin mud-slips, with a little galena in pockets in the central part of the vein. The spar is bedded in layers from 18 in. to 2 ft. thick, the planes probably resulting from compression of the walls. Along the SE. wall the fluorspar is somewhat purplish, and the wall itself shows some replacement by zinc-blende in fine grains. The vein narrows downward. At the floor of the drift it is only 6.5 ft. wide, and 15 ft. further on it has contracted to 2.5 feet.

Fifty feet from the incline the vein presents the following structure from foot to hanging-wall: white fluorspar, 8 ft. 2 in.; calcite and limestone breccia, 1 ft.; limestone, 8 in.; purple fluorspar, 1 ft. 6 in.; limestone sheet, 1 ft. Another section in this drift, also beginning with the foot-wall, which is calcite-seamed, shows: calcite, very irregular, 1 to 3 in.; limestone, 6 to 8 in.; calcite, 4 in.; limestone, with short calcite-seams in center, 6 in.; calcite, 2 in.; limestone, 9 in.; dark, rotten fluorspar, 6 in.; zinc carbonate, 4 in. Such alternations of rich and lean portions of the vein occur for the 500 ft. or more of the southwest course of the drift. The vein is more or less sinuous, with a general strike N. 33° E. The ore-shoots, over 3 ft. in width, vary from 30 to 90 ft. in length. Bordering these are narrower shoots, from 5 to 40 ft. long, between which

are comparatively barren stretches of from 20 to 40 ft., with or without narrow shoots. In this 70-ft. and the 110-ft. level these shoots pitch 28° to 33° SW. from the horizontal. The vein throughout shows a persistent central slip, sometimes deflected to one side and sometimes filled with barite. This would be expected, as the greatest width of the ore-channel would offer the path of least resistance. The slickensided grooves parallel the pitch of the ore-shoots.

At the 110-ft. level the conditions are much the same, except that the fluorspar is more solid and shows no muddy shale for the first 275 ft., when a horse takes up the greater part of the vein for 50 ft. This horse is much brecciated, and cemented with calcite. and occasionally with fluorspar. Beyond the horse, fluorspar of good quality is again obtained.

At 142 ft. down the incline St. Louis limestone sets in as the foot-wall. At the 160-ft. level the vein shows fluorspar as good as that above, and of a similar character, except that the pitch of the ore-shoots seems to have been reversed, and there is occasionally a little flint, only partly replaced by fluorspar, on the foot-wall side. At a depth of 210 ft. the vein was less than 3 ft. wide, and carried much zinc-blende. At 220 ft. in depth the vein carried 3 ft. of fluorspar; the foot-wall here changed to semi-oölite and the hanging-wall from Ste. Genevieve oölite to the flinty, compact limestone of the St. Louis formation. At the time of my last visit, the incline had been sunk to about 210 ft. in depth, at which point the vein was less than 3 ft. wide, and carried much zinc-blende. Below this pinch it was expected that white fluorspar would again be encountered. In the old workings, for 1,200 ft. to the NE., the vein shows a similar character, with a width not exceeding 7.5 feet.

The mechanical equipment of the Memphis mine is simple, but sufficient for present requirements. The fluorspar is dumped from skips into a short, narrow, steep trough with a screen bottom, into which a stream of water discharges, washing the lumps, which fall into a bin at the end, while the gravel and the small material passing through the screen go to a log-washer. Both the lump and the gravel are hand-culled and sorted, and wheeled to different dumps, according to grade. This mine has produced in a month, with 11 men at work under and above ground, about 800 tons of merchantable fluorspar.

About 700 ft. NW. of the Memphis vein is an outcrop of Birdsville quartzite, which marks the master-fault of the Columbia fault-zone. This fissure carries ore-shoots running high in lead and zinc, as well as others, consisting of fluorspar alone or together with galena and calcite. The fault at this point shows a displacement of about 500 ft. Ohara limestone forms the SE., and Birdsville quartzite the NW. wall. The fault dips NW. and is paralleled by the Memphis group of veins.

II. Origin of the Deposits.

The formation briefly described above is the result of earth-movements curving the strata into a monoclinal fan-fold (the uplift end of which is perhaps connected with the Rough Creek anticline), followed by the action of forces attending the intrusion, and producing dikes and sills of the basic igneous rocks, mica-peridotite and pyroxene-lamprophyre, which caused the formation of local domes. The fault-zones and fissures were the results of tension incidental to the doming, in parallel and complementary simultaneous sets. Into these faults and fissures thermal waters brought fluorine, etc., in the silico-fluorides of lead, zinc, barium, etc., which were precipitated by various reagents—the fluorine by calcium, and the metals by hydrocarbons, hydrogen sulphide, etc.

III. Fluorspar-Mills.

The mill of the Kentucky Fluorspar Co., typical of its class, may be seen at Marion, preparing for market the product of the district. The spar, received by wagon from the mine, is dumped on piles according to grade and character, such as red gravel, white gravel, "leady" fluorspar, "zinky" fluorspar, calcite fluorspar, and various grades of "lump-ore." Pickers select the best "white lump," and wheel it into the warehouse. Different grades are run through the mill at different times. The cleaning-machinery comprises a small jaw-crusher, a set of rolls, and a trommel-screen, and two seven-compartment jigs, the latter serving to wash the spar thoroughly and also to rid it, as far as possible, of attendant lead, zinc, calcite, etc. Lead comes from the first cell and hutch, middlings from the second, and fluorspar from the next four, while tailings partly go over, and partly are caught in the last cell. The last hutch-product

and the middlings are returned for another run through the mill. The spar intended for grinding is elevated to a drier and moved by a conveyer into a bin, from which it passes by gravity to a pair of buhr-mills, which reduce it to a fineness of 65 or 85 mesh. It is then bagged or barreled for shipment. For fuller details of mining, mine-timbering, cleaning, grinding, costs, etc., reference may be made to my report to the Kentucky Geological Survey.[1] In the same bulletin will be found sections of the Memphis mine and a map of the Memphis group of veins.

IV. The Use of Fluorspar in the Metallurgy of Iron and Steel.

The value of fluorspar in the manufacture of iron and steel depends upon its ability to form two types of allied slag-products, whereas an acid or a basic flux forms only one. It first forms volatile acid products, in which fluorine is the controlling factor, and then basic slag-forming compounds, in which calcium is the characteristic constituent. Fluorspar is chiefly serviceable, therefore, in the "basic" processes of steel-manufacture, because both the products formed by it are destructive of acid furnace- or converter-linings. Basic slags, basic furnace-linings, and gently-oxidizing conditions are necessary if the most favorable results are to be obtained from the use of fluorspar in the making of iron and steel. The thorough purification of iron requires the elimination of silicon, sulphur, and phosphorus, and with these the fluorine of the spar forms fluorides, which are volatile acid compounds, while its calcium forms silicates, sulphides, and phosphides, which are basic slagging compounds. These reactions, and those which occur between limestone and silicon, sulphur, and phosphorus, are as follows:

(1) $2\ CaF_2 + SiO_2 = 2\ CaO + SiF_4$ and

(2) $2\ CaO + 2\ SiO_2 = 2\ CaSiO_3$; or

(3) $2\ CaF_2 + 3\ SiO_2 = 2\ CaSiO_3 + SiF_4$.

(4) $3\ CaF_2 + 4\ S = 3\ CaS + SF_6$.

(5) $5\ CaF_2 + 7\ P = 5\ CaP + 2\ PF_5$.

(6) $3\ CaCO_3 + 3\ SiO_2 = 3\ CaSiO_3 + 3\ CO_2$.

[1] *Bulletin No. 9, Kentucky Geological Survey,* Kentucky Fluorspar Deposits, with Notes on the Production, Mining, and Technology of the Mineral in the United States (1907).

(7) $3\ CaCO_3 + 3\ S = 3\ CaS + 3\ CO + 6\ O$.

(8) $3\ CaCO_3 + 3\ P = 3\ CaP + 3\ CO + 6\ O$.

It will be seen from equations (1) and (2) that silica combines with calcium fluoride and sets free two molecules of calcium oxide, which will satisfy two additional molecules of silica, the result being that of equation (3). In the case of sulphur and phosphorus there is a direct interchange of constituents. The reactions attending the joint use of fluorspar and limestone would be expressed by combining equations (3) and (6), (4) and (7), (5) and (8).

The volatilization by fluorine, as shown by these reactions, of one-fourth to one-third of the difficultly-fusible constituents of slag, and the resulting increase in lime-content, seem to present the following distinct metallurgical advantages: (1) The slag is more basic, fusible, and liquid; (2) fusion is effected at a lower temperature, which, together with the heat-yield from the formation of fluorides, reduces fuel-consumption; and (3) the concentration of the slag increases the metal-output.

Fluorspar has a very limited value, if any, in assisting in the removal of carbon or manganese. An excess of this flux over the quantity required to flux the silica, sulphur, and phosphorus, would alloy graphite and manganese with the iron and reduce silica to silicon; and fluorides would be formed which, in the reaction with the hydrogen of the furnace-gases, would be reduced to metals and hydrofluoric acid. Such an employment of fluorspar requires very careful manipulation; but it may be made to give, as desired, either pure iron, iron of slightly altered qualities, or a distinct alloy.

In quantitative effect, fluorspar has a distinct advantage over other basic fluxes—about two to one as compared with calcium carbonate, the cheapest of them. Moreover, by reason of the volatilization of one-third of the impurities, it forms only two-thirds as much slag as limestone. But it costs eight or ten times as much as limestone; and large quantities of it would produce effects opposite to those desired. Hence the best present practice is to use as a flux in iron and steel metallurgy a comparatively large proportion of limestone and a small proportion of fluorspar (in order to secure the peculiar effects of the latter). This practice is not expensive. For example, if limestone costs $0.50 and fluorspar $5 per ton, the use of 3 per

cent. of fluorspar in the flux would make the cost per ton of flux 63.5 cents, instead of 50 cents, or only 9 per cent. of increase in the cost of the flux for each 1 per cent. of fluorspar.

The substitution of other basic fluxes for part of the limestone will not interfere with the action of the fluorspar; but some of them, like dolomite, require a greater percentage of fluorspar to lower their melting-point.

The net fluxing-value of a crude fluorspar may be determined with sufficient accuracy by deducting the silica plus 2 units of basic impurities or 1 unit of calcium fluoride for each unit of silica. Thus, in a material containing 91 per cent. of calcium fluoride, 2 of silica and 7 of calcium carbonate, alumina, iron oxides, etc., the 2 units of silica could be regarded as practically neutralized by 4 units of the basic fluxes, leaving as the net flux 91 of calcium fluoride and 3 of basic compounds; whereas, if the material contained 91 calcium fluoride and 9 silica, it would be necessary to deduct 9 units of the fluoride to satisfy the silica, leaving as net flux 82 per cent. of calcium fluoride.

The standard methods for the analysis of fluorspar, as described and discussed by Brush, Penfield, Richards and others, need not be stated here, since they are easily accessible to students. But I may call attention to a method proposed by Randolph Bolling[2] as sufficiently accurate and rapid for use in open-hearth steel-works using fluorspar as a flux. Such practical methods are useful not only for determining the available proportion of flux, as a guide to the smelter, but also as a check on the sellers of the crude spar.

Fluorspar is commercially obtainable in four grades. Stated in percentage, the first carries from 96 to 98 calcium fluoride, with not more than 2 silica; the second, 90 fluoride, with less than 4 silica; the third, 80 fluoride, with a maximum of 12 silica; and the fourth, about 60 fluoride, with a maximum of 15 silica.

Concerning these grades, it may be added that the first, by reason of its cost, is little used in iron- and steel-works; the second is the most available for that purpose, though in some cases it might be economical to require (and pay for) a material considerably lower in silica than 4 per cent. The third is likely

[2] *Iron Age*, vol. lxxviii., No. 19, p. 1258 (Nov. 8, 1906); also, *Bulletin No. 9, Kentucky Geological Survey*, p. 158 (1907).

to give (after due allowance for the basic constituents) about 78 per cent., and the fourth a minimum of 60 fluoride with 10 per cent. of available basic flux. The economic limit for the purchaser is about 8 per cent. of silica; and the third and fourth grades should be purchased only on the basis of their net fluxing-value, as determined by the method given above.

The proportion of fluorspar which can be used with advantage is thus seen to be a variable, dependent upon the price of the crude material, the process in which it is employed, and the impurities of flux, fuel, and ore. The calculations of the furnace-manager should be made accordingly. It may be said, however, that where pure limestone is the main flux, from 1.5 to 8 (on the average, 3) of fluorspar to 100 of limestone is sufficient to secure the main advantages of the compound flux; for magnesian limestone, from 15 to 30 per cent. of the fluoride may be required.

The fluorspar-limestone flux may be useful in the manufacture of pig-iron, wrought-iron, crucible-steel, basic Bessemer and open-hearth steel, iron- and steel-alloys, and ordinary and malleable-iron castings. In rare instances, and in small quantities, it may be serviceable in the acid steel-processes also.

In blast-furnace practice, it has been, as yet, but little used, for the reason that its advantages are not generally understood, and its cost has been regarded as prohibitory. As has been shown above, however, the additional expense of using a certain percentage of fluorspar is not great in comparison with the economic and technical benefit thus secured; and its use may therefore reasonably be expected to increase. It is specially advantageous in the smelting of highly-siliceous ores, and for the purpose of "thinning" a too-limy slag—for which it is used by the Illinois Steel Co. It may be blown as a powder through the tuyeres, or intimately mixed with the charge.

It should be of service in smelting iron-ores in the electric furnace, since it lowers the temperature of fusion. It is known to be serviceable for this purpose in other electric-furnace operations, as in the Lungwitz zinc-process, in making alundum, carbolite, etc.

In the Bessemer process, because of the strongly oxidizing conditions, fluorspar is little used. But even here, it appears, according to Howe, to assist, by melting the lime rapidly, in

making an effectively basic slag, with which phosphorus combines readily during the early part of the operation.

It is in the basic open-hearth process that American fluorspar is mostly used, especially in plants which produce steel for rails, tubing, or castings, for which the highest quality of open-hearth steel is required. In this process, fluorspar is used to facilitate the liquefaction or thinning of the slag (especially to help melt the flux or limestone), thus reducing, by about half, the time required for the melt. It is only to be used when the limestone is soft and white-hot, floating at the top of the bath, usually about two hours prior to the completion of the heat. If used sooner, it makes the slag too thin. If too much limestone is charged, fluorspar is helpful in converting it to slag quickly. An excess of fluorspar thins the slag too much, with harmful results, such as too great lowering of the temperature; the rapid oxidation of the carbon; the excessive oxidation of iron, which causes losses of metal, and also thins the slag; and the too-rapid oxidation of other impurities (such as phosphorus and sulphur), which gives them opportunity to become again reduced and alloyed with the metal. Phosphorus is more readily oxidized than sulphur, and also returns earlier to the metal.

The percentage of fluorspar to that of limestone in this process varies from zero to 8 per cent., with 3 per cent. as an average. Where the slag is sufficiently thin, or if sufficient scale (iron oxide) has been used, no fluorspar at all may be required. If the slag is thick and melts slowly, a little fluorspar is shoveled in. An intelligent furnace-man requires less fluorspar than a careless one. The requirement of fluorspar per furnace is about 100 tons per year. "Gravel" fluorspar is used, usually of the second grade.

The procedure in open-hearth practice, according to a private communication from J. W. L. Kerr, is as follows: Magnesite and dolomite are first put in as furnace-lining. Afterwards 6.5 tons of limestone and 25 or 35 tons of scrap-iron are charged and heated together. At the right heat, the melted pig-iron necessary to make a total charge of 60 or 70 tons is added. Lake Superior hematite is then added until the carbon is reduced from 0.18 to 0.08 per cent. Pieces of limestone come to the surface in blocks; and fluorspar is used in amounts varying from 200 to 1,100 lb., or about 0.15 to 0.8 per cent. per

charge, both to break up the limestone blocks and to reduce sulphur and phosphorus.

At the South Chicago Works of the Illinois Steel Co., the practice, according to a private communication from George L. Danforth, Jr., is to use from 0 to 15 lb. of fluorspar (6 lb. on the average) to 200 lb. of limestone per ton of basic open-hearth steel produced, the iron charged being half pig, half scrap. No actual weights are taken; and the gravel fluorspar is shoveled in as needed when the limestone is at the right heat.

For a charge of 50 tons of cheap scrap-iron, the use of 2.5 per cent. of fluorspar is said to give steel as fine as that produced from the best pig.

Carr proposes the addition of 13 lb. of fluorspar with 300 lb. of limestone to 1,227 lb. each of pig-iron and steel scrap for each ton of basic open-hearth cast-steel produced.

Fluorspar is a valuable flux in the preparation of iron- and steel-alloys. Its value depends on its use in excess in conjunction with a highly-basic flux in reducing to an elementary state carbon, manganese, silicon, chromium, nickel, etc., and alloying them with the iron or steel, as previously explained. In the ordinary blast-furnace, ferro-silicon, containing as much as 10 per cent. of silicon, can be produced from any siliceous ore in this manner. Likewise, spiegeleisen, low or high ferro-manganese, or metallic manganese may be produced. In the last case, the flux consists of alumina, lime, and fluorspar.

The objections that have arisen in some quarters to the use of fluorspar in foundry-practice are due to the ignorance of its capabilities and manner of use. The dealers who sell fluorspar as a flux under high-sounding names, making extravagant claims as to its effect in extremely small quantity, and charging correspondingly high prices for it, are chiefly to blame for this. That its use alone has proved unsatisfactory is not surprising. When it has been used in conjunction with limestone, the failures have been due to an insufficient amount of total flux, usually with the additional error of an improper proportion of fluorspar and limestone in the mixture. This explains the unsatisfactory results of the Foundryman's Laboratory tests, reported by N. W. Shed, in which it was sought to reduce sulphur and phosphorus without making allowance for either the fluorspar or the limestone which would necessarily flux part of the silica

of both the coke and the pig-iron—a neutralization which left no fluorspar available to reduce the sulphur or the phosphorus.

The present foundry-practice is to use pure limestone (calcium carbonate) for flux in the cupola, 100 lb. of limestone to a 2-ton charge, and either no fluorspar at all or only such small quantities of it as are necessary to help bring about a quick melt of the limestone. If no pure limestone is available, dolomite is used, and in conjunction with this flux fluorspar is indispensable. In many foundries, especially in the small ones, only a single two-hour heat is made per day, so that the time consumed is immaterial. In the large foundries continuous heats are demanded, with an output of at least 15 tons per hour. The use of fluorspar reduces the length of time required for each melt. Where fluorspar is used with dolomite, only one-half as much time is required as for dolomite without fluorspar. The type of cupola used makes little difference. A typical practice in this respect is that of the Crane Co., Chicago, which, after repeated attempts to get pure limestone for flux, settled upon the use of fluorspar and dolomite in the ratio of 25 or 30 per cent. of fluorspar to 100 per cent. of dolomite. For example, a 2-ton charge, consisting of pig-iron and scrap, with sand-covered gates, would require 25 lb. of fluorspar and 85 lb. of dolomite for flux, the two being mixed together and shoveled into the cupola. Analyses of typical fluorspar and dolomite used are as follows:

	SiO_2.	P.	Fe_2O_3+ Al_2O_3.	CaF_2.	$CaCO_3$.	$MgCO_3$.	PbS.	ZnS and $ZnCO_3$.
Fluorspar,	1.37	0.006	0.50	96.75	....	....	0.08	1.69
Dolomite,	1.08	0.007	0.68	....	53.50	45.19	......	......

Such fluorspar has 94 per cent. of available calcium fluoride, for some fluorspar is necessary to flux the lead and zinc as well as the silica. A fluorspar is specified for this use that contains less than 2 per cent. of silica, and the dolomite must not contain a greater amount. This means that only No. 1 fluorspar can be used. Lump is purchased, as it is likely to contain less silica. After the fluorspar reaches the foundry it is broken into pieces not larger than the size of an egg, since larger pieces would be likely to strike the lining and, combining with it, destroy it. The cupolas have the usual fire-brick linings, which, if the precaution is observed, are but slightly if

at all affected. The sulphur is not likely to be reduced materially by the use of fluorspar if there be less than 1 per cent., but if there be more the reduction of sulphur will be marked.

Aside from its use in the cupola, a small percentage of ground fluorspar, placed at the bottom of the ladle, serves to slag impurities, which rise to the surface as a heavy mass, and, after stirring to insure perfect mixture, are skimmed off. According to R. C. Hills, gray iron so treated produced not only a softer iron, but when molded into bars and broken on a testing-machine, showed 11 per cent. greater breaking-strain than bars made from the same pig not so treated. For malleable iron, similar treatment showed a more malleable iron with an increased tensile strength (55,000 to 60,000 lb.), and an increase in elongation (4 to 5 per cent.) over ordinary malleable iron. Frogs made out of this material cracked to a far smaller extent than ordinarily.

The use of fluorspar in the iron and steel industries is extending. At present, the percentage of fluorspar consumed is one-half of one per cent. of the quantity of limestone flux used in America; but there has been a small but steady increase in its use during recent years. This is chiefly due to the extension of the basic open-hearth steel manufacture, which now consumes fully 80 per cent. of the output of American fluorspar. Of this quantity, more than 90 per cent. is in the "gravel" form. About 5 per cent. of the total American fluorspar-consumption is used by other branches of the iron and steel industries, chiefly in the manufacture of car-wheels and ordinary and malleable-iron castings. For the first purpose, lump is chiefly used; for the last two, either gravel or ground fluorspar.

Out of about 800 iron- and steel-plants, only about 65 are known to use fluorspar, chief among the users being all the larger and more important of the basic open-hearth plants. Out of hundreds of foundries, only about 150 have used fluorspar. With the assured practically inexhaustible supplies of Kentucky fluorspar, accessible for transportation by rail and water to the large iron- and steel-centers, it is believed that its use could be generally extended in the iron and steel industries within the limits herein set forth, without material increase in cost, and frequently with material improvement of product.

Vanadium-Deposits in Peru.

BY D. FOSTER HEWETT, PITTSBURG, PA.

(New Haven Meeting, February, 1909.)

THE scope of this paper is the description of two districts in Peru in which deposits of vanadium have been found, and the consideration of much laboratory-work that I and others have done to determine the nature of these deposits. It is with regret that I present some of the data in a state of evident incompletion, particularly the analyses of a few of the minerals, which leaves some doubt as to whether the true nature of the minerals is understood. I make no apology, however, since in this case, as in many others referring to ore-deposits, conclusive opinions cannot be stated until a great amount of work has been done, and the deposit has been thoroughly exploited. I believe that the data and the tentative conclusions here presented indicate a unique and interesting condition which may assist others in work along similar lines; and further, what is of more practical importance, stimulate a search for a metal which, though long considered rare, is probably so merely because proper care has not been taken to recognize it.

An excellent bibliography on the occurrence of vanadium in nature is given by F. W. Clarke in his paper, Data of Geochemistry;[1] and in the preparation of this paper I have consulted many of the references contained in the Bibliography of the Chemistry of Vanadium, by M. Moissan.[2] A brief review of the subject having special reference to the metallurgy of vanadium is contained in the small book by P. Nicolardot.[3]

One of these vanadium-deposits has already been briefly

---

[1] *Bulletin No.* 330, *U. S. Geological Survey* (1908).

[2] *Chimie Minérale,* vol. iii. (1905).

[3] *Le Vanadium* (1905).

described,[4] and analyses of some of the minerals have been published.[5]

Both of the vanadium-districts are situated in the Department of Junin; Yauli, the less important, being in the Province of Tarma, and Quisque (Minasragra) in the Province of Pasco. The Peruvian Central railroad (Ferrocarril Central Del Peru) passes through the first district, and the second is accessible by

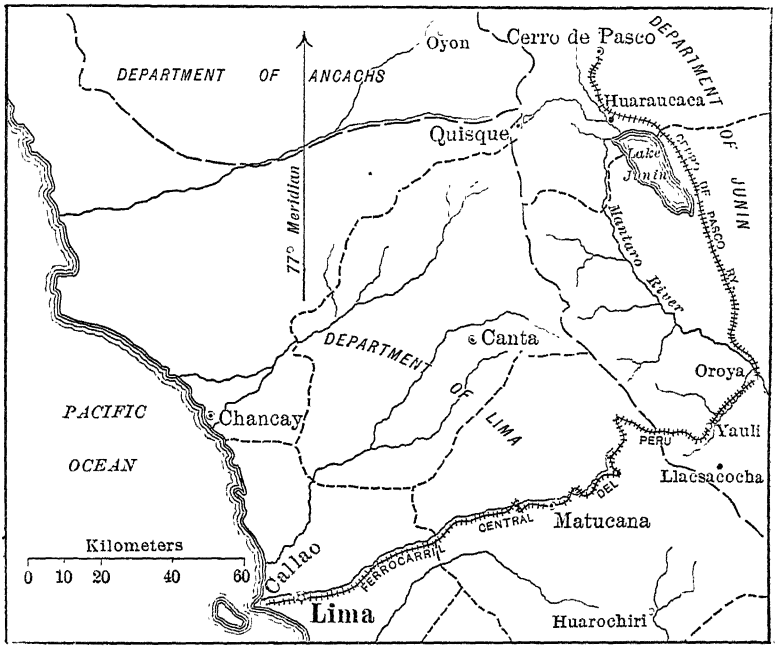


FIG. 1.—MAP OF A PORTION OF PERU, SHOWING LOCATION OF VANADIUM-DEPOSITS.

horseback from Fundicion on the Cerro de Pasco railroad, as shown in the map, Fig. 1.

Though within 80 miles of the Pacific ocean, both districts are on the eastern slope of the main range of the Andes, and

[4] El Vanadio de Minasragra, by José J. Bravo, in *Boletin de la Sociadad de Ingenieros*, Lima, vol. viii., No. 8, pp. 171 to 185 (August, 1906); and A New Occurrence of Vanadium in Peru, by Foster Hewett, *Engineering and Mining Journal*, vol. lxxxii., No. 9, p. 385 (Sept. 1, 1906).

[5] The Vanadium Sulphide, Patronite, and Its Mineral Associates from Minasragra, Peru, by W. F. Hillebrand, *Journal of the American Chemical Society*, vol. xxix., No. 7, p. 1019 (July, 1907); also, The Present Source and Uses of Vanadium, by J. Kent Smith, *Trans.*, xxxviii., 698 (1908).

the drainage in each case is into the Mantaro river, the waters of which enter the Ucayali, the main branch of the Amazon.

Topographically, the districts are similar, though the relief in the Yauli region is stronger. The features depend upon geological structure rather than variety of rock; faults and folds in the sedimentary rocks being common.

The main range of the mountains, lying approximately N. 30° W., is well defined. Summits often attain elevations of 18,500 ft., and passes are seldom lower than 16,000 ft. The main range is succeeded to the east by similar though lower parallel ranges. The intervening valleys, when narrow, resemble the U-shaped glacial valleys of the Rocky mountains, but when broad (up to 20 miles) they are low in relief, and contain many lakes and marshes.

There are two distinct seasons—the wet, which begins during November and lasts until May; and the dry, which comprises the remaining months. During the wet season precipitation occurs daily and regularly, even to the hour. Above 14,000 ft., in latitude S. 12°, it is invariably snow, and is often confined to the higher ranges. Except in sheltered places, the snow melts quickly below 17,000 ft., so that it can be understood that erosion by water in altitudes of from 14,000 to 17,000 ft. is a minor feature, a consideration that has great bearing upon the deposit in the Quisque district.

During the dry season precipitation may occur, but it is irregular, and streams in the upper valleys become dry to such an extent, in fact, that the herds of sheep and llamas must be taken to the lower valleys for pasture. Severe electrical storms, unaccompanied by rain, are common during the dry season.

A description of the Yauli district will precede that of the more important Quisque district, for the reason that the first enables a more thorough understanding of the second, as well as the fact that this was the order in which I examined them. It was a coincidence that the Quisque deposit was discovered while I was examining the Yauli district.

The Yauli District.

Historical.—Though not announced until 1894,[6] the presence of vanadium in the so-called "anthracite" (asphaltite) of the

[6] *Journal of the Chemical Society of London*, vol. lxx., pt. 2, p. 252 (1896); also, Torrico y Mesa, *Boletin de Minas*, Peru, vol. x., No. 12, p. 94 (Dec. 31, 1894).

Yauli region was known in 1892. Theretofore, the asphaltite had been held under denouncement, and exploited with the view to utilizing it as fuel. In 1895 the Llacsacocha deposits were acquired by a French company, whose principal resources were some nearby silver-mines (Andaychagua). This company attempted to utilize the vanadium contained in the asphaltite, but, with the impoverishment of the silver-veins in depth, the company became bankrupt and ceased operations in 1899. It is reported that several tons of vanadiferous ashes were sent to France for treatment, but beyond this very little asphaltite was mined. No work has been done since 1899.

Several of the less-important deposits in the northern por-

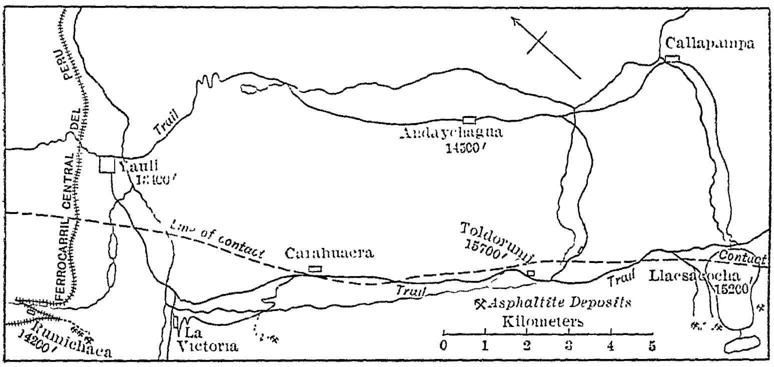


FIG. 2.—MAP OF THE DISTRICT OF YAULI, PERU, SHOWING LOCATION OF THE ASPHALTITE-DEPOSITS.

tion of the district have been developed to a slight extent, and the asphaltite locally used for fuel.

Geology.—Fig. 2, embracing an area about 5 by 12 miles, gives the location of known occurrences of asphaltite.

The broken line on the map marks the contact between folded Jura-Trias and Cretaceous sedimentary rocks on the southwest, and more recent eruptive rocks on the northeast, which contain the silver-veins of Andaychagua and the copper-silver veins of Yauli. The famous Carahuacra silver-mine, and several less important, are in the contact-zone.

The Jura-Trias rocks are coarse conglomerates and sandstones, and the Cretaceous is represented by thin-bedded gray and green shales and limestones, the latter predominating, illustrated in Fig. 14. In the vicinity of Toldorumi the intru-

sion of the eruptive rocks on the northeast has brought up a narrow belt of slates.

The veins of asphaltite occur in a well-defined belt in the sedimentary rocks, following the stratification and approximately parallel, therefore, to the contact mentioned above. The stratification varies from N. 15° W., with a dip of 75° NE., in the vicinity of Rumichaca, to N. 50° W., with a dip of 60° SW., in the vicinity of Llacsacocha. (*Cocha* is the Indian (Quichua) name for "lake," so Llacsacocha "lake" would be superfluous.) Asphaltite-veins have been found over a distance of 15 miles.

The veins occur as lenses, varying from 0.5 in. to 22 ft. wide, the maximum known length of a lens being about 500 ft. These veins are not necessarily confined to one bedding-plane, and may not only break through from one to another, but there may be asphaltite to the extent of 5 ft. in width, filling each of two or three contiguous bedding-planes, so that there appear to be three veins separated by single layers of shale from 3 to 10 in. thick. There is evidence of movement subsequent to the intrusion of the asphaltite, which is shown by the pencillate structure of the asphaltite, and numerous faults of small throw.

The asphaltite is black, lustrous, and softer than ordinary bituminous coal. The hardness does not appear to be affected by the amount of ash present. The material lacks completely any evidence of clay-bands and the columnar structure common to most bituminous coals. It breaks clean from the walls, though the country-rock adjoining often contains a large amount of carbonaceous matter. No pyrite was observed in the asphaltite at any of the exposures.

The mode of occurrence of the asphaltite is essentially the same at the various exposures marked on the map, so that only the most important, that on the north side of Llacsacocha, will be described.

The vein has been developed by five tunnels, aggregating about 2,000 ft., within a vertical elevation of 300 ft. The tunnels are from 150 to 500 ft. long. Table I., a record of samples from Tunnel No. 2, serves to explain in detail the variation in the width of the vein, and the amount of vanadium contained in the asphaltite

TABLE I.—*Samples of Asphaltite from Tunnel No. 2.*

Sample No.	Distance.	Width.	Ash.	Vanadic Oxide.	Factor.
	Feet.	Feet.	Per Cent.	Per Cent.	
1	5	5	1.24	1.13	0.0115
2	32	5	13.62	0.91	0.0106
3	66	6	1.35	1.03	0.0105
4	105	5	1.30	1.16	0.0119
5	145	7	6.27	1.40	0.0150
6	182	8	20.14	0.71	0.0090
7	198	6	25.52	0.64	0.0090
8	227	5	13.58	0.82	0.0095
9	246	6	1.31	0.91	0.0093
10	267	7	0.98	0.86	0.0089
11	307	4	46.90	0.67	0.0129
12	366	4	20.93	1.14	0.0145
13	409	4	3.32	1.25	0.0130

Table II. shows the amount of ash and of vanadic oxide contained in the samples from the various tunnels.

TABLE II.—*Analyses of Samples of Asphaltite from Various Tunnels.*

Place.	No. of Samples.	Ash.	Vanadic Oxide.	Factor.
		Per Cent.	Per Cent.	
Tunnel No. 1	5	6.78	1.17	0.0125
Tunnel No. 2	13	12.05	0.96	0.0109
Tunnel No. 3	9	7.60	1.07	0.0115
Tunnel No. 3½	6	2.55	1.12	0.0116
Tunnel No. 4	4	1.75	1.15	0.0116
General average	...	7.33	1.06	....

The series of proximate analyses in Table III. were made in the hope of recognizing some relation between the possible metamorphism of the asphaltite and the proximity of the eruptive contact, but such does not appear to be the case. Nor was it found that there was any apparent relation between physical properties and constitution.

It is apparent that the percentage of ash in the asphaltite from the Yauli district varies considerably. Further, the amount of vanadic oxide in the residue after burning varies between wide limits. This appeared to suggest that the vanadium was independent of the mineral matter in the asphaltite and that, therefore, its origin might be traced to the hydrocarbon portion.

The last column in Tables I., II., and III. gives a factor de-

TABLE III.—*Analyses of Various Samples of Asphaltite.*

Sample No.	Moisture.	Volatile Matter.	Free Carbon.	Sulphur.	Ash.	V_2O_5.	V_2O_5 in Ash.	Factor.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
1...	0.98	9.41	84.45	5.52	5.16	0.68	13.1	0.0072
2...	....	....	....	....	2.02	1.30	64.3	0.0133
3...	....	....	....	....	2.26	1.43	43.9	0.0148
4...	....	....	....	....	8.15	0.82	10.0	0.0090
5...	1.50	9.84	86.52	7.24	2.14	1.32	61.7	0.0135
6...	0.52	8.26	90.58	4.28	0.64	0.49	77.5	0.0050
7...	....	....	....	....	1.78	1.34	75.3	0.0136
8...	0.66	15.23	63.31	4.07	17.80	trace	....	....
9...	7.83	49.02	31.15	4.54	12.00	0.28	2.3	0.0032
10...	....	....	....	....	9.38	0.89	9.5	0.0098
11...	0.32	11.16	85.02	6.47	3.50	0.94	27.0	0.0098
12...	3.62	48.59	44.28	44.59	3.51	0.54	16.4	0.0056
13...	0.08	20.34	78.57	1.87	1.01	0.19	19.27	0.0020

1 Yauli district, 1 mile south of Llacsacocha.
2 Yauli district, ½ mile south of Llacsacocha.
3 Yauli district, ½ mile south of Llacsacocha.
4 Yauli district, ¼ mile south of Llacsacocha.
5 Yauli district, southeast of lake.
6 Yauli district, end of Tunnel No. 1.
7 Yauli district, from surface, 1,000 ft. north of lake.
8 Yauli district, hole near Carahuacra.
9 Yauli district, opening near Rumichaca.
10 Yauli district, opening near Rumichaca.
11 Yauli district, opening near Rumichaca.
12 Quisque district, Minasragra.
13 Page district, Indian Territory, U. S. A., McClure's Mine.

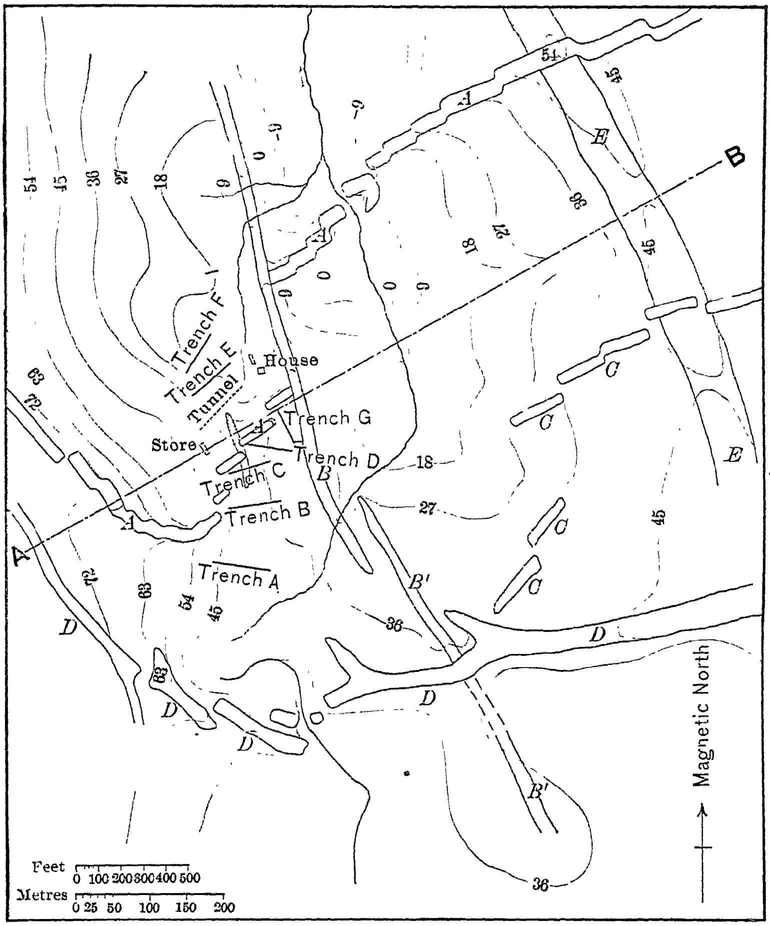
rived by dividing the percentage of vanadic oxide in the asphaltite by the percentage of hydrocarbon, considering all combustible matter as hydrocarbon. It appears to me that this factor is constant enough to be considered an index of the solubility of vanadium (probably as the sulphide) in the hydrocarbon. This matter will be referred to later.

About 100 ft. east of the asphaltite-vein there is an outcrop of a dike of devitrified perlitic obsidian about 50 ft. wide. It appears to have been intruded along one of the bedding-planes. There are no contact-phenomena other than a noticeable hardening of the shales, though the dike appears to have been the cause of the formation of a zone of pyrite more or less replacing the shale in the foot-wall of the asphaltite-vein.

QUISQUE DISTRICT.

Historical.—About Nov. 20, 1905, a party of Indians, who had been in the mountains searching for coal, brought to Señor

Antenor Rizo Patron, metallurgist at the Huaraucaca smelter, on the Cerro de Pasco railway, samples of a material thought



Contours are given in meters above arbitrary datum-line. *A*, *B*, *B′*, *C*, *D*, and *E* are faulted dikes.

FIG. 3.—TOPOGRAPHIC MAP OF MINASRAGRA (LA QUIMICA MINE).

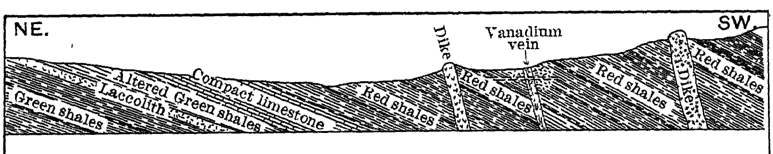


FIG. 4.—SECTION ALONG LINE **A–B**, FIG. 3.

to be coal. The material came from a prospect near the crest of the main Cordillera, within the concession of the *hacienda*

Quisque, the Indians having been led in this direction in the hope of finding an extension of the coal-fields of Oyon, 20 miles north. The prospect had been located for coal no less than three times previously, but was abandoned in each case when it was found that the material contained a large amount of sulphur. Under these old locations the prospect became known as Minasragra.

Upon analysis, the material was found to contain a high percentage of vanadium, which fact was subsequently confirmed in the laboratory of the Corps of Engineers in Lima. I was able to see the prospect in January, 1906, and as engineer for the American company which purchased the property, I have subsequently made two visits to the prospect.

Geology.—The geology can best be explained by reference to Figs. 3, 4, 5, 6, and 7.

The area under consideration lies along the western limb of a broad anticlinal in Jura-Trias and Cretaceous rocks, similar in character to those exposed near Yauli, and over a large area along the axis of the mountains in this portion of Peru. The section shows the series in this locality to be composed of green shales, thin-bedded limestones, and red shales. The red shales are succeeded to the west by a great thickness of limestones, of which the main range is composed. The vanadium-deposit occurs entirely within the red shales.

Igneous activity in the form of the intrusion of dikes, laccoliths, and domes, has been a feature throughout a large area in this portion of the Province of Pasco, and these forms have been controlling factors in determining the local topography. It is evident from the map, Fig. 3, that this locality has been a unique center of this activity, there being no less than four systems of dikes, and the entire area being probably underlain by a laccolith.

The following rocks have been identified:

Dike B′.—A typical trachyte in an advanced state of decomposition. Contains red orthoclase, hornblende, and much secondary magnetite and calcite. Biotite and quartz are absent.

Laccolith E.—Dolerite, much altered. Contains oligoclase (?); short prisms of augite; idiomorphic olivine, slightly altered; laths of hornblende; pyrite, calcite, original and secondary magnetite.

Dike C.—Typical andesite, comparatively fresh. Contains phenocrysts of oligoclase; laths of hornblende, and some biotite in a ground-mass of feldspar. This rock bears some resemblance to dikes *A* and *D*, though it shows more alteration.

Dikes A and D.—Typical quartz-porphyry, as shown in Fig. 13. Contains predominating plagioclase in phenocrysts; orthoclase, generally corroded; quartz as rounded grains, and as comparatively perfect di-hexagonal pyramids; biotite, in a ground-mass of feldspar fragments. The feldspars from four

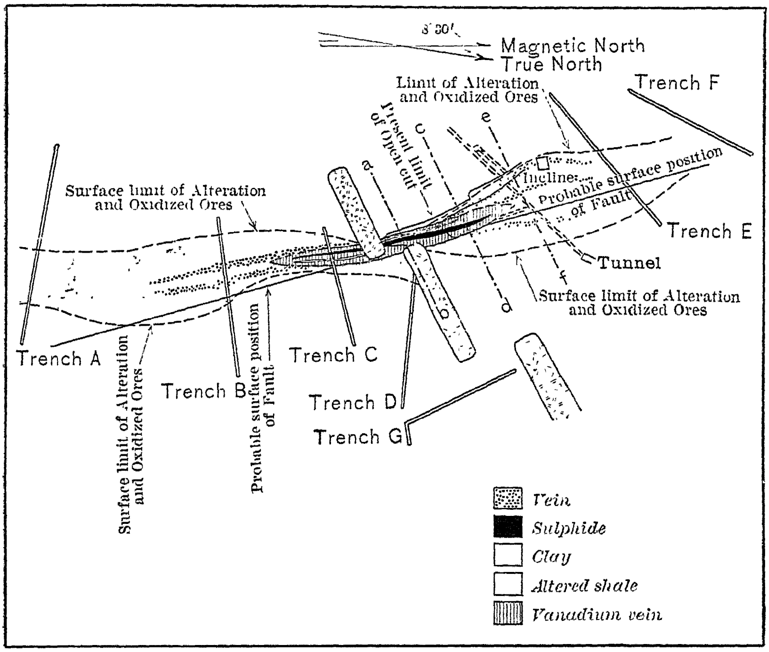


FIG. 5.—MAP OF LA QUIMICA MINE, MINASRAGRA.

specimens are crushed, due to stress. There can be no doubt that dikes *A* and *D* are derived from the same magma, and probably are contemporaneous.

Dike B.—Diabase, without olivine, slightly decomposed. Contains decomposed hornblende; plagioclose; much magnetite, both original and secondary; no biotite, quartz, or olivine.

The following order of intrusion is deduced from the extraordinary number of intersections of the dikes, and the unique faulting: 1, dike *B′*; 2, laccolith *E*; 3, dike *C*; 4, dikes *A* and *D* (contemporaneous); 5, faulting; 6, dike *B*.

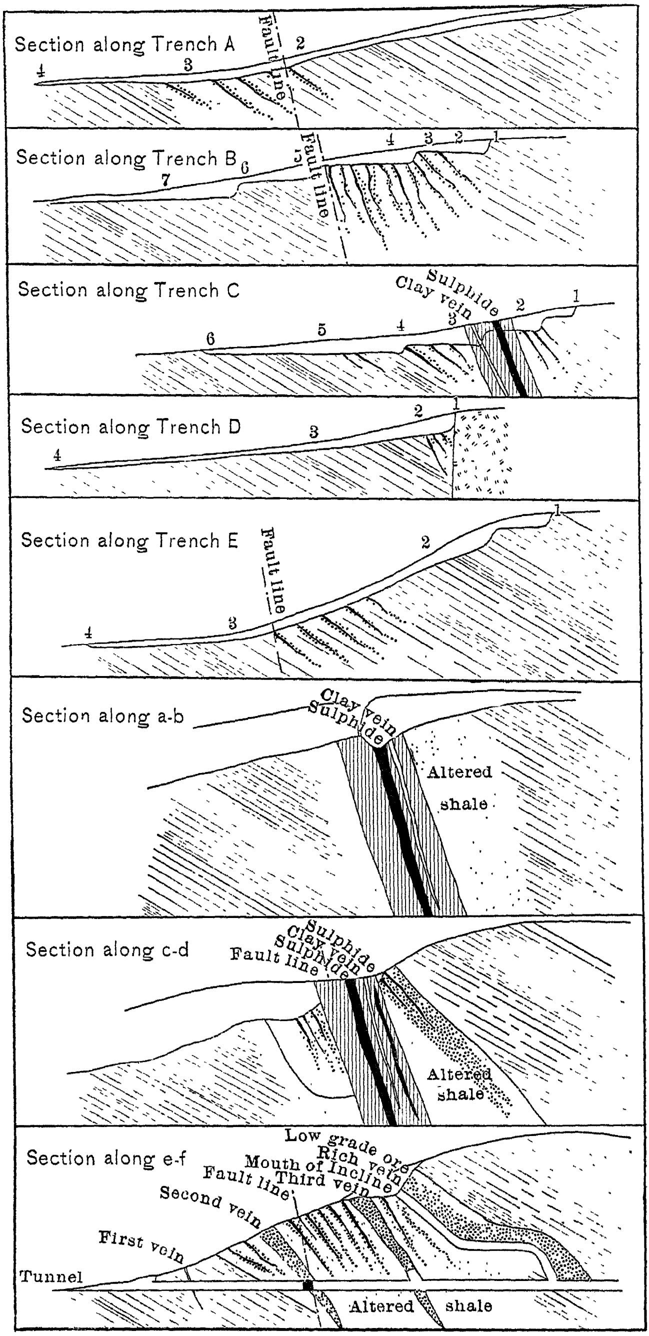


FIG. 6.—SECTIONS ALONG TRENCHES AND ON ARBITRARY LINES NORTH OF DIKE *A*.

From the evidence of stress during solidification in dikes *A* and *D*, and the lack of it in dike *C*, it appears that the faulting began when *A* and *D* were yet semi-molten, but when *C* was completely solidified. Furthermore, *A* and *D* are fresh rocks, and *C* shows a considerable amount of decomposition, which could scarcely be attributed to atmospheric weathering.

A study of the map, Fig. 3, will show the faulting to have been of an extraordinary nature. Proceeding both eastward and westward from the crest of the ridge, where the laccolith outcrops, the faulting has been progressively southward. Further west from the vanadium-deposit the faulting is again reversed. It appears, therefore, that the area under consideration has

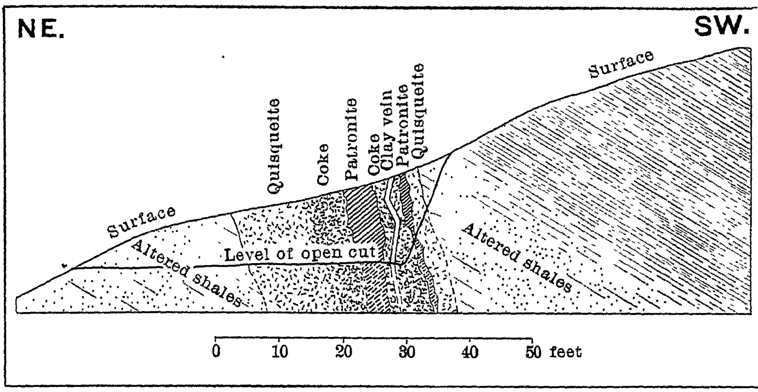


Fig. 7.—Section through Vanadium-Deposit North of Dike.

undergone faulting due to stresses resembling torsion in the earth's crust.

Of the five intrusions, the laccolith is the only one which appears to have strictly followed the stratification of the rocks. Though dikes *B* and *B'* have a strike corresponding to that of the shales, they are vertical, as shown in the section. Unusual contact-phenomena were not recognized in connection with any of the dikes, though the green shales overlying the laccolith are hardened.

The Vanadium-Deposit.

The vanadium-deposit proper is a lens-shaped mass composed principally of three distinct constituents, which occupies one of the faults of the dike *A*. The maximum width of this mass is about 28 ft., and though the length is partly concealed,

it cannot exceed 350 ft. The strike is about N. 20° W., and the dip is 75° W.

The three materials in order of the relative amount present are:

Quisqueite; a black, lustrous hydrocarbon; hardness, 2.5; sp. gr., 1.75; fracture, conchoidal. Name derived from the *hacienda* in which the deposit is located.

Coke; a dull black, vesicular hydrocarbon; hardness, 4.5; sp. gr., 2.4; fracture, conchoidal. This coke contains globules of quisqueite.

Patronite; a greenish-black mineral; hardness, 2.5; sp. gr., 2.65 to 2.71; fracture, uneven. Name derived from that of Señor Antenor Rizo Patron, who first recognized that the mineral contained vanadium.

The following minerals, found in very small amounts, were recognized under the microscope:

Bravoite; a reddish-yellow mineral in patronite. Composition, (Fe, Ni) S_2; hardness and specific gravity not known. Name given by Dr. Hillebrand in honor of Señor José J. Bravo, who has described the deposit.

An undetermined silicate mineral resembling halloysite.

Analyses of these minerals made by Dr. Hillebrand are:

	Patronite. Per Cent.	Quisqueite. Per Cent.	Coke. Per Cent.
Sulphur, sol. in. CS_2,	4.5	15.44	0.64
Sulphur, combined,	54.29	31.17	5.36
Carbon,	3.47	42.81	86.63
Hydrogen,	....	0.91	0.25
Nitrogen,	....	0.47	0.51
Oxygen, by difference,	....	5.39	4.64
Water, at 105°,	1.90	3.01	None
Ash,	....	0.80	1.97
Vanadium,	19.53		
Iron,	2.92		
Nickel,	1.87		
Silica,	6.88		
Titanic oxide,	1.53		
Alumina (phosphoric acid),	2.00		

Also small amounts of ferric oxide, manganese, chromic oxide, and alumina.

For comment upon these analyses, and the action of various reagents on the materials, as throwing light on their constitution, see the original article by Dr. Hillebrand.[7]

[7] *Journal of the American Chemical Society*, vol. xxix., No. 7, p. 1019 (July, 1907).

The specimens of patronite which were analyzed by Dr. Hillebrand were not as pure as some I have since been able to obtain. Analyses of pure material yield from 19.3 to 24.8 per cent. of vanadium, and the siliceous matter is lower than shown in the analyses by Dr. Hillebrand. I believe that the siliceous matter and possibly the carbon may be considered impurities, and that patronite approaches in composition a compound of vanadium and sulphur, which can best be represented by the formula $V_2S_5 + nS$.

The following extract from an article by Kay[8] on The Sulphides of Vanadium has some bearing upon this phase of the question:

"At a temperature of about 400°, vanadium trisulphide takes up two additional atoms of sulphur, forming the pentasulphide, V_2S_5. For the preparation of this compound the trisulphide is mixed with one-third its weight of sulphur (purified by carbon bisulphide), the mixture very finely powdered and placed in a strong narrow tube. The tube, after filling with dry carbon dioxide, is temporarily closed by a cork provided with a narrow capillary tube (to prevent access of oxygen), and the tube is then sealed off as near as possible to the surface of the mixture. The sealed tube is next heated in a hot-air oven for three hours, to a temperature of about 400°. . . . The product, after washing with hot carbon disulphide to remove any free sulphur present, leaves a residue of the pure pentasulphide.

It is inadvisable to use a large excess of sulphur in the preparation of this sulphide, for although the pentasulphide is produced, it retains from one to two per cent. of sulphur very tenaciously, and the removal of this is a matter of considerable difficulty."

I am unable to say whether the compositions of quisqueite and of the coke vary from the analyses given, for these are the only complete analyses that have been made. By analogy to many other natural hydrocarbons, I believe that the composition may vary within a small range from that given.

The nature of patronite and its relation chemically and genetically to quisqueite in coke will be made clear by a close study of the photomicrographs, Figs. 8 to 13. No photomicrographs have been made of the purest patronite, but an examination of several sections under a magnification of 200 diameters, after polishing and etching with caustic potash, showed a perfectly homogeneous surface, whose relief only varied with the degree of attack by the reagent.

[8] *Journal of the Chemical Society of London*, vol. xxxvii., p. 738 (1880).

The photomicrographs show patronite, quisqueite, coke, bravoite, and the siliceous mineral, in varying proportions. In Figs. 8, 9, and 10, quisqueite is the black material, generally in the form of globules, though sometimes only a thin film lines the cavities, as in Fig. 10. The coke is black, but much duller in luster than quisqueite, and is found in the sections as an irregular net-work resembling a sponge. Patronite fills all of that portion not occupied by the coke and quisqueite, and is the light material showing the greatest relief in Fig. 10. Bravoite, which is of very irregular occurrence, is shown only in Fig. 12 as the unetched crystal in the darker ground-mass of patronite. The white segregations in Fig. 8 are the siliceous mineral referred to above.

Taken in connection with the larger structural relations shown in Fig. 7, it is clear that the materials are segregations from a mass which was probably originally homogeneous. Quisqueite was the first material to segregate. The larger portion was crowded to the walls, though a small amount was caught as globules in a viscous mass remaining. It probably segregated through insolubility in the remaining mass, rather than on account of any difference in melting-points, for it appears to have been viscous after the segregation of the coke. Moreover, it was probably unstable at the temperature then existing, this being suggested by the globules, which are more or less coked. The coke segregated after the quisqueite, and though nearly pure carbon, must have had the consistency of a paste, for it was able to form solid, fairly homogeneous masses.

Before the solidification of the patronite took place, there must have been a disturbance throughout the mass, due probably to its upward movement. This is shown by a crushed condition in the coke, Fig. 11, and to a less degree in the quisqueite, Fig. 9.

Patronite was the last mineral to solidify, and from its associations appears to have been the eutectic of the mass. It fills all of the cracks and spaces between the globules of quisqueite, Figs. 9, 10, and 11, and the walls of the cavities. The fact that patronite is of the nature of a eutectic serves to explain why its composition does not accord with the usual valency of vanadium, as eutectics, though generally constant in composition, are seldom compounds to which a formula may be assigned.

2-in. eye-piece; 1.5-in. objective; magnification, 15 diameters; oblique illumination. Silicate mineral, white. Quisqueite, black globules. Coke, dark net-work. Patronite, gray ground-mass.

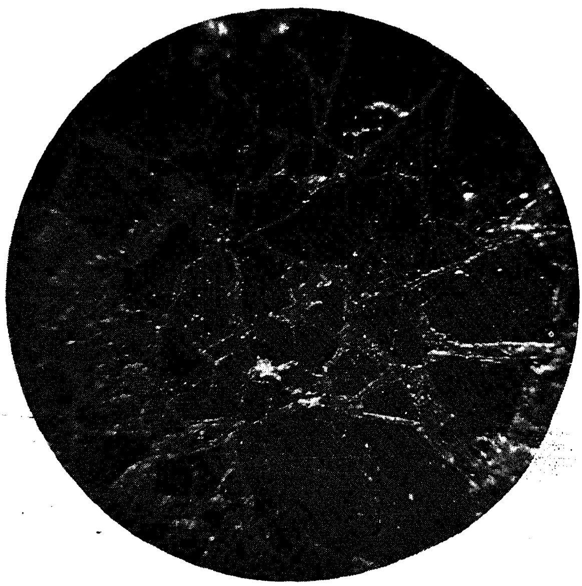
FIG. 8.

2-in. eye-piece; 1.5-in. objective; magnification, 15 diameters; oblique illumination. Quisqueite, black globules. Coke, dark net-work. Patronite, light ground-mass.

FIG. 9.

1-in. eye-piece ; 1.5-in. objective ; magnification, 20 diameters ; oblique illumination. Quisqueite, black, lining cavity. Coke, dark. Patronite, light groundmass.

FIG. 10.

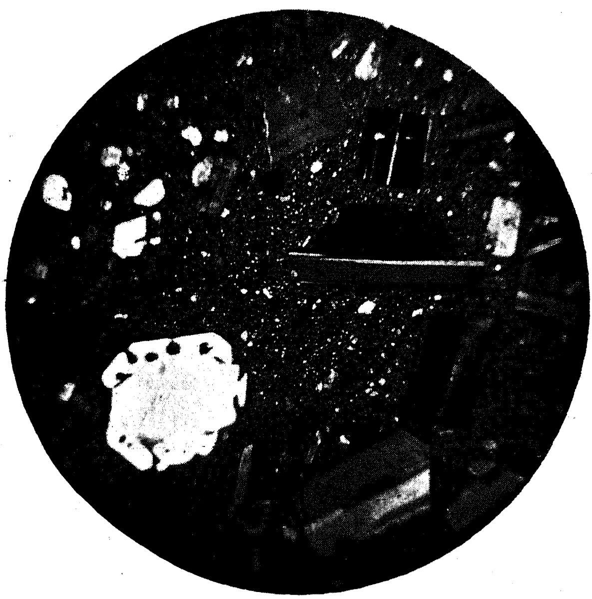


1-in. eye-piece ; 1.5-in. objective ; magnification, 20 diameters ; oblique illumination. Coke, dark fragments. Patronite, gray, filling cracks.

FIG. 11.

2-in. eye-piece; 1.5-in. objective; magnification, 15 diameters; oblique illumination.

FIG. 12.—CRYSTAL OF BRAVOITE (Fe, Ni) S_2, IN GROUND-MASS OF PATRONITE.



1-in. eye-piece; 1.5-in. objective; magnification, 20 diameters; crossed nicols.

FIG. 13.—DIKE *D*, WEST END: QUARTZ-PORPHYRY.

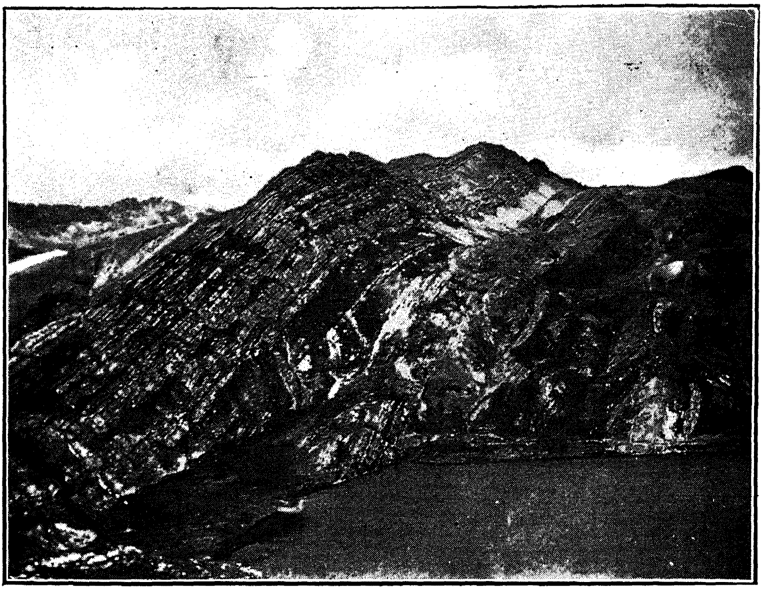


FIG. 14.—LLACSACOCHA AND VICINITY, SHOWING OPENINGS ON ASPHALTITE-VEIN. ELEVATION OF LAKE, 15,200 FEET.

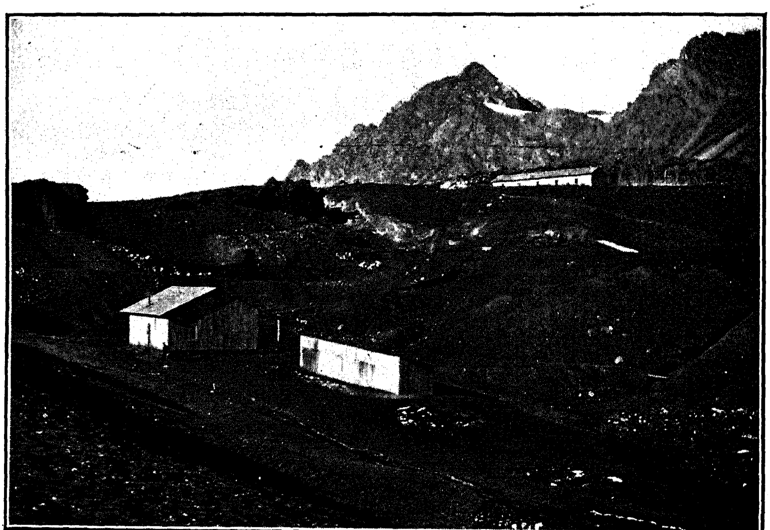


FIG. 15.—VIEW SHOWING OPEN-CUT OF LA QUIMICA MINE (QUISQUE DISTRICT), DEPARTMENT OF JUNIN, PERU.

Patronite appears to have had the peculiar property, under the conditions of temperature and pressure which existed at the time of the intrusion of the mass, of being able to permeate the porous country-rock, even to the degree of saturating it. This condition is found in the hanging-wall of the mass, and the impregnation is generally in the neighborhood of a veinlet of quisqueite, suggesting that this latter material follows a fissure which was also the means of access of the patronite.

At some period, after the mass had become solid, a conclusive movement reopened the line of fault, rending the mass from end to end, as shown by the clay-filled fissure, Figs. 5 and 7.

The red shales are not only crushed, due to the faulting prior to the appearance of the mass, but much bleached, due probably to the action of the sulphurous vapors which accompanied the intrusion. The portions of dike *A* near the fault are also much decomposed, the feldspars being completely altered to kaolin. Explorations from the tunnel at a depth of about 50 ft. from the surface show the shales to be much silicified, and to be replaced by irregular zones of fine, granular pyrite.

Effects of Oxidation.—The development at the time of the last inspection, August, 1907, consisted of an open-cut, a tunnel about 270 ft. long, and six trenches, of which five cross the zone of intrusion, Figs. 3 and 5. This work facilitates a more thorough understanding of the nature and size of the deposit, and of the effects of oxidation, than was theretofore possible.

It is evident from the contours on the map, Fig. 3, that dike *A* determines the drainage of much of the area. Through the oxidation of the sulphide, patronite, there have been formed solutions of vanadium which, passing north and south along the west side of the mass, have precipitated oxide ores of vanadium. In some cases the solutions have almost entirely replaced portions of the shales, and at other places the vanadium minerals have been precipitated in the cracks and open spaces in the crushed zone. The mineralization of the shales has generally taken place along well-defined zones, though in places rich shoots of ore have been formed which pass into lean ore in the walls. Furthermore, where upon the surface there are found large areas of low-grade ore with occasional rich streaks, there are found at the tunnel-level (maximum vertical depth about 120 ft.) the same richer portions confined within barren material.

Two minerals of approximately definite composition have been found upon the surface. Both are hydrated oxides of vanadium, and though partial analyses only have been made, they appear to be new species. Dr. Hillebrand is inclined to regard them as salts, not as oxides or acids. The first, which will be called the "red oxide," has been analyzed by Dr. Hillebrand, with the result given below. It occurs typically as globular aggregates with radiated structure, though amorphous material is common. The color is deep brownish-red to red; streak, red; sp. gr., 2.30 to 2.48; hardness, 2.5. The second, which will be called the "brown oxide," has not been analyzed in detail. One analysis that I made shows: loss on ignition, 19.0; and vanadic oxide, 72.50 per cent.

The brown oxide is found along the outcrop in irregular amorphous masses, none having a tendency towards more definite structure. It is dark brown in color; streak, dark brown, sectile; hardness, 2.0; sp. gr., 2.30. It has the peculiar property of swelling and disintegrating when placed in water.

TABLE V.—*Analyses of Red Oxide and Green Oxide.*

	Red Oxide. Per Cent.	Green Oxide. Per Cent.
Vanadic oxide, V_2O_5	67.60	57.33
Hypovanadic oxide, V_2O_4	trace	4.76
Molybdic oxide,	2.82	3.28
Silica,	1.17	0.57
Titanic oxide,	} 3.31	0.07
Alumina,		......
Phosphoric oxide (?),		......
Ferric oxide,		19.53
Lime,	4.30	0.70
Magnesia,	?	trace
Water,	20.81	13.89

At a depth of 8 or 10 ft. from the surface, the red and brown minerals are replaced by a mineral having a greenish-black color, occurring under very much the same conditions. The mineral is generally amorphous, though along openings or water-courses it appears velvety, which appearance, under the microscope, is resolved into aggregates of acicular crystals, much resembling a common form of malachite. The specific gravity of the mineral is 2.52. Both analyses in Table V. were given by Dr. Hillebrand in a personal communication to me.

I suggested to him that the "red oxide" may be ortho-

vanadic acid (H_3VO_4), which has not been known in nature heretofore, and that the "green oxide" may be a combination of V_2O_4 and V_2O_5 in approximately constant proportions. Analyses of three samples of this "green oxide" gave the following results:

	Vanadium.	Vanadic Oxide. V_2O_5.	Hypovanadic Oxide. V_2O_4.
	Per Cent.	Per Cent.	Per Cent.
1, ...	16.46	23.58	5.20
2, ...	24.38	44.49	8.22
3, ...	40.17	54.19	15.76

Dr. Hillebrand has replied to these suggestions as follows:

"You see that the proportion of V_2O_4 to V_2O_5 in the second of my analyses (green oxide) does not bear out your suggestion, when compared with your ratios, that it is a definite proportion. There is, indeed, no reason to expect other than a varying ratio between the V_2O_4 and V_2O_5 under the circumstances of occurrence. Every conceivable proportion might exist between the initial stage of oxidation of the patronite and the final stage represented by the red matter.

"It strikes me that we must, from the nature of the occurrences, have mixtures of minerals in both cases. Until we know how the iron (especially the iron in the second analysis), molybdenum, V_2O_4, and lime are combined, it is useless to attempt to fix a formula for the V_2O_5 compounds, although quite permissible to indicate probabilities.

"It appears that the ortho-, meta-, and pyro-vanadic acids are incapable of free existence. If formed momentarily, they pass at once into hexavanadic acid, $H_4V_6O_{17}$, of which the orange incrustation (see below) seems to be a salt. The acid itself is, however, unstable, and in its further decomposition gives rise to separation of V_2O_5. I may add that the acid itself is not known in the free state, but its existence is assumed from the behavior of its solution in water. In view of the above, I am at present inclined to think that the red substance of which you furnished me specimens (red oxide), and which you think to be orthovanadic acid, may be a mixture of some calcium salt and of V_2O_5, possibly also of an iron salt. Against this, however, is the large amount of water shown by my analysis, an amount about equal to that found in the orange incrustation, but with apparently smaller amounts of bases The presence of molybdenum in the material first analyzed, of which very little is present in the orange incrustation (see below), renders more difficult the forming of any conclusion regarding the nature of your supposed H_3VO_4. The same holds true for your $H_4V_2O_7$ (green oxide), which, on account of the large amount of iron present in it, I am disposed to regard as essentially an iron vanadate. I am afraid that we had better not attempt to form any positive conclusions regarding the composition of these two products which are so evidently mixtures."

It is probable that there is a gradual transition from vanadic oxide compounds on the surface to those of hypovanadic and vanadic oxides below the surface, though the large amount of iron shown in Dr. Hillebrand's analysis makes his suggestion that the material which he analyzed was iron vanadate undoubtedly correct.

Solution of vanadium compounds is in process continually, and waters flowing from the tunnel show a considerable amount of vanadium in solution. The waters, percolating through the walls of the tunnel, deposit an orange incrustation, after analysis of which Dr. Hillebrand comments as follows:

"I have finished an approximate analysis of the crystalline orange incrustation on one specimen which you sent me quite a while ago. This orange incrustation is apparently a hydrous calcium salt of hexavanadic acid, that is, $Ca_2V_6O_{17} + x\,H_2O$. The x is between 10 and 11. Water is about 22 per cent., of which 14 per cent. comes off over sulphuric acid, if the exposure is prolonged for several months, after which time the exposure at 100° C. gives rise to no further loss. The mineral then loses little water until the temperature has gone up quite a bit, but at or below 250° becomes wholly dehydrated.

"The percentage of calcium oxide is about 12.8 per cent., that of V_2O_5 about 65.4 per cent. There is also a very small amount of insoluble matter that has been deducted in deriving these figures."

Table VI. is a record of analyses of samples prepared by quartering one wheel-barrow load, from trench B and from the open-cut on the north side of the dike, and gives an idea of the extent of the replacement of the shales by vanadium compounds:

TABLE VI.—*Analyses of Samples from Trench B and the Open-Cut.*

Trench B.

Average Over.	Vanadic Oxide. Per Cent.
Section A, 29 ft., east end,	1.96
Section B, 35 ft., adjoining,	3.26
Section C, 35 ft., adjoining,	12.86
Section D, 25 ft., west end,	3.26

Open-Cut.

No. of Sample.[a]		Thickness. Ft.		Vanadic Oxide. Per Cent.
16	Richer zone.	6		57.62
18		8		48.51
20		7		25.72
22		6		6.35
24		8		10.25
26		4		7.97
17	Poorer zone overlying.	7	above No. 16	5.53
19		8	above No. 18	8.13
21		10	above No. 20	3.58
23		4	above No. 22	1.46
25		4	above No. 24	4.36
27		3	above No. 26	26.36

[a] Samples taken at 10-ft. intervals, horizontally.

The tunnel cuts this same zone of replacement at a depth of 120 ft. along the dip of the shales. The following samples show the diminution in the percentage of vanadium.

	Vanadic Oxide. Per Cent.
No. 35, tunnel, average of 20 ft.,	9.61
No. 36, tunnel, average of 9 ft.,	7.81
No. 37, tunnel, average of 4.5 ft.,	6.51

This impregnation, or replacement of the shales, can, of course, only extend to the depth of ground-water circulation, which in this immediate vicinity will probably not exceed 100 ft. vertically (about 200 ft. on the dip of the shales).

Production.—Within the period from June, 1906, to January, 1909, there has been produced and shipped to the United States about 1,800 tons of oxidized ores, containing about 20 per cent. of vanadic oxide; also the product obtained from roasting about 400 tons of sulphide ore, patronite.

CONCLUSIONS.

The occurrence of vanadium in hydrocarbons is not confined to Peru, for it has been announced by Kyle in the ash from a coal (asphaltite ?) found in the province of Mendoza, Argentine Republic; also from other localities mentioned in Clarke's Data of Geochemistry.[9] I have found it in the ash from an asphaltite from Page, Indian Territory (now Oklahoma). The material contained 1.10 per cent. of ash, of which 0.19 per cent. was vanadic oxide. A review of the original articles describing the above-mentioned occurrences has convinced me that all of the materials are very closely related, if not almost identical, in nature and occurrence. In other words, they all appear to be asphaltites containing an appreciable amount of sulphur.

Investigators of the subject seem agreed with the opinion expressed by Eldridge in his article, entitled The Formation of Asphalt Veins,[10] that asphaltites are derived from petroleum. Richardson,[11] after a most thorough study of the subject from a chemical standpoint, shows that all asphalts contain sulphur,

---

[9] *Op. cit.*

[10] *Economic Geology*, vol. i., No. 5, p. 437 (Mar.-Apr., 1906).

[11] On the Nature and Origin of Asphalt, by Clifford Richardson, *Bulletin* No. 1, Barber Asphalt Paving Co.; also, *Journal of the Society of Chemical Industry*, vol. xvii., No. 1, p. 13 (Jan. 31, 1898).

and that the hardness appears to depend upon the amount present.

"Asphalts are distinguished by the large amount of sulphur they contain, and it is to its presence that many of the important characteristics, and perhaps in part, the origin of this form of bitumen is due . . .

"The harder and least soluble portion always contains the larger part of the sulphur. It seems, therefore, that sulphur is the effectual hardening agent of natural asphalts, in the same way that it is of artificial asphalts which are produced by heating a soft natural bitumen with sulphur. . . .

"It seems justifiable, therefore, to suggest that where certain mineral oils, composed of alicyclic hydrocarbons, originate under such circumstances as to be subjected to conditions favorable to condensation and polymerization, or to the action of sulphur or sulphates, asphalt will be formed, not necessarily immediately, but in the course of time."

Further, a method of preparing the sulphides of vanadium, described by Carnot,[12] has some bearing:

V_2S_3 is prepared by heating an oxide or a chloride of vanadium in hydrogen sulphide, or vapor of sulphide of carbon. This sulphide, heated in a current of hydrogen, furnished the bisulphide, V_2S_2, and in the vapor of sulphide of carbon, at 400°, the pentasulphide.

Taking these observations into consideration, the occurrence of vanadium in asphaltites appears to depend upon three factors:

1. Vanadium, as oxide, disseminated through a rock of a fair degree of porosity.
2. Impregnation with a hydrocarbon to a greater or less degree.
3. A source of sulphur or sulphureted vapors.

The first of these conditions undoubtedly often exists over large areas. In the two districts in Peru under consideration, the two remaining conditions have undoubtedly been brought about by the intrusion of the dikes. It would seem, therefore, a safe forecast, that most asphaltites containing more than 2 per cent. of sulphur contain vanadium.

The Quisque deposit may now be interpreted as an extreme phase of differentiation from asphaltite, the intrusion of the dikes probably having had the effect of successively concentrating the vanadium. Further, the unique climatic conditions account fully for the formation of the large aureole of oxidized ores. Had there been erosion by water even to a slight degree,

[12] *Traité d'Analyse*, p. 778 (Paris, 1904).

the oxidized minerals would have been carried away, instead of being permitted to accumulate in the porous country-rock.

With the exception of the analyses made by Dr. W. F. Hillebrand, of Washington, to determine the composition of the minerals, and those in Table VI., which were made by J. O. Handy, of Pittsburg, Pa., I have done all the analytical and research-work given in the present paper. Cordial acknowledgment is hereby made to Dr. Hillebrand, whose analytical work has been of invaluable assistance. Acknowledgment is also made to Señor Felipe de Lucio, mining engineer in charge of development of the property, for unqualified co-operation in every phase of the investigation and work on the property.

The Residual Brown Iron-Ores of Cuba.

BY C. M. WELD, NEW YORK, N. Y.

(New Haven Meeting, February, 1909.)

ATTENTION has been turned recently to the exploration and development of certain large blanket-deposits of brown iron-ore in Cuba. The most conspicuous of these to-day, and the one upon which the most light has been shed, is the Mayari deposit, situated about 15 miles south of Nipe bay. Here the Spanish-American Co. has sole control over 18,500 acres of ore-bearing lands, reported by its engineers to contain 500,000,000 tons of ore. The necessary plant and equipment, with docks and railways, is now under construction for the early marketing of this ore. A similar deposit, and undoubtedly the next to be exploited, is the ore-field at Moa bay, where from 13,000 to 15,000 acres of ore-lands, immediately adjacent to the shores of an excellent harbor, have been generously covered by numerous mining-claims, practically all controlled by four large interests. This deposit is now estimated to contain approximately 350,000,000 tons, on the basis of dried ore ready for shipment, a figure which may be increased when the western limits of the ore-deposit have been more accurately defined. Other deposits of the same type, but smaller and less accessible, are those at Cubitas, situated from 12 to 15 miles north of Camaguey city,

and at Taco bay and Navas, points lying a few miles west of Baracoa. The area of the Cubitas deposit is said to be 6,000 acres, and the yield of ore is estimated at 150,000,000 tons. The Baracoa deposits are less well known, but preliminary estimates have placed their joint ore-reserves at 40,000,000 tons.

Accepting the above tonnages as reasonably correct, we conclude that the deposits enumerated give promise of adding

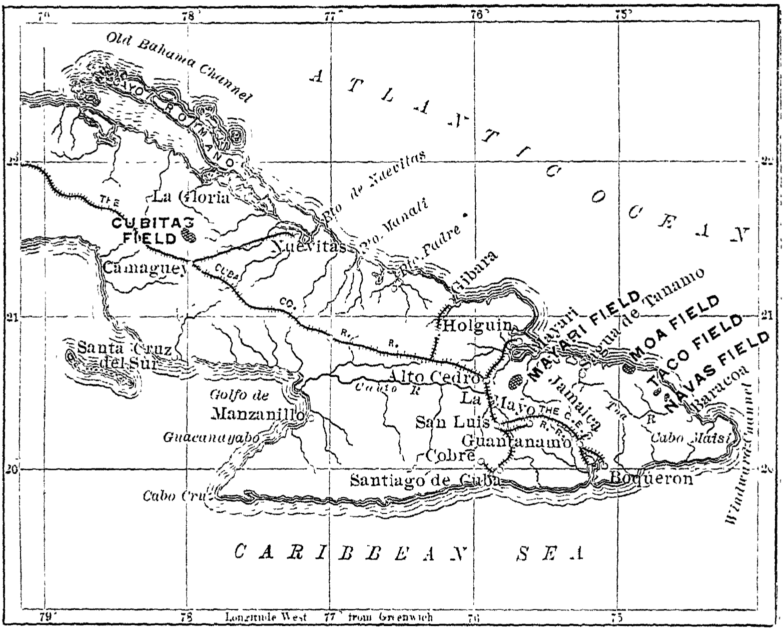


Fig. 1.—Map of Province of Santiago de Cuba, Showing Deposits of Brown Iron-Ore.

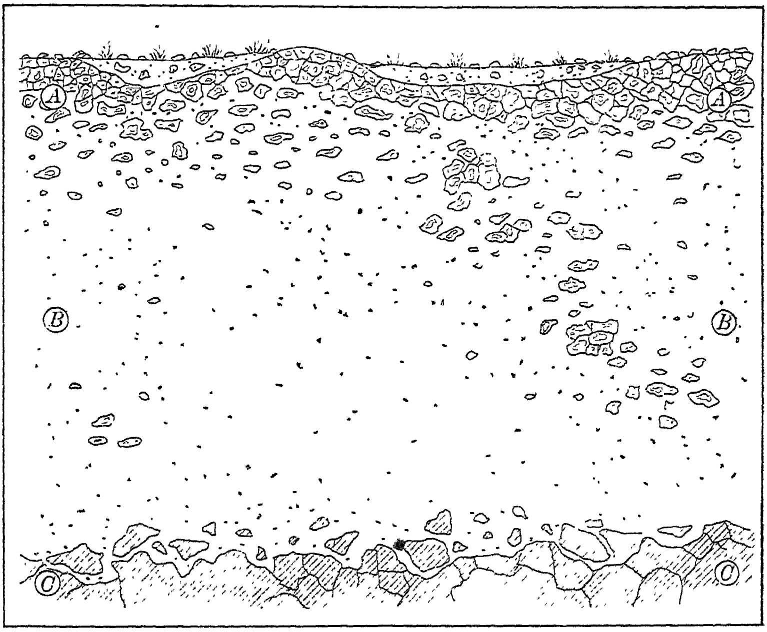
about 1,000,000,000 tons of iron-ore to the world's supply; they have, therefore, to be considered in any attempt to forecast the future of the iron and steel industries.

The map, Fig. 1, shows the approximate location of all the deposits mentioned. An illustrated description of the Mayari deposit, and the proposed plant and equipment for its exploitation, has already been published;[1] also, additional information, together with a brief reference to the Moa deposit.[2] These two papers are largely commercial in their attitude. A. C.

[1] *Iron Age*, vol. lxxx., No. 7, pp. 421 to 426 (Aug. 15, 1907).
[2] *Ibid.*, vol. lxxxi., No. 15, pp. 1149 to 1157 (Apr. 9, 1908).

Spencer, in his paper entitled, Three Deposits of Iron Ore in Cuba,[3] gives valuable and interesting information along more purely scientific and technical lines regarding the deposits at Mayari, Moa, and Cubitas.

While the subject of the present paper is therefore not altogether new, it has appeared to me that certain features concerning the character and probable genesis of the iron-ore deposits



A. Recemented capping of brown ore, from 1 or 2 up to 10 ft. thick. Frequently absent.

B. Clay-ore, red, yellow, or brown in color, from 7 or 8 up to 50 or 60 ft. thick. Contains disseminated nodules and pellets of brown ore, at times agglomerated in the form of beds or layers.

C. Serpentine bed-rock.

Fig. 2.—Idealized Vertical Section Showing Nature of Occurrence of Residual Iron-Ore in Cuba.

have not yet been brought out, and it is with this in view that the paper has been prepared.

The deposits under discussion possess essential characteristics in common. They occur as residual mantles of enormous surficial extent, with a thickness occasionally as great as from 50 to 60 ft., but more commonly varying from 10 to 20 ft. The

[3] *Bulletin No.* 340, *U. S. Geological Survey*, pp. 318 to 329 (1908).

underlying rock is serpentine. The ore, which extends from the grass-roots to bed-rock, is for the greater part a homogeneous, tenacious, clay-like material, red to yellow to brown in color. The transition between ore and the comparatively unaltered serpentine bed-rock is as a rule fairly abrupt. Within the clay-ore are found disseminated nodules and pellets of brown ore ranging apparently through all the hydrated forms from limonite to turgite; hematite also is present, and at times magnetite. These concretionary forms increase in abundance towards the top of the ore-bed, where they frequently appear as recemented masses of spongy brown ore, occasionally of large dimensions, forming beds or layers within the clays. At many places a capping or crust, as it were, of recemented brown ore of considerable extent is found at the immediate surface.

Fig. 2 represents an idealized section of the ore-beds from grass-roots to bed-rock. This sketch has been constructed from numerous observations in the field, and presents graphically the conditions above described.

Table I. gives representative analyses from four different fields. That for Mayari is taken from the article cited above,[4] while the others are of samples taken in connection with private examinations and not before published.

TABLE I.—*Analyses of Brown Iron-Ores of Cuba.*

	Mayari.	Moa.	Taco.	Navas.
Fe	46.03	46.75	46.23	42.48
SiO_2	5.50	1.71	2.06	3.01
Al_2O_3	10.33	11.60	2.16	6.12
Cr	1.73	1.81	2.07	2.39
TiO_2	*a*	0.14	*a*	*a*
P	0.015	0.031	0.021	0.032
H_2O	13.62	13.15	*a*	*a*

[a] Element not reported.

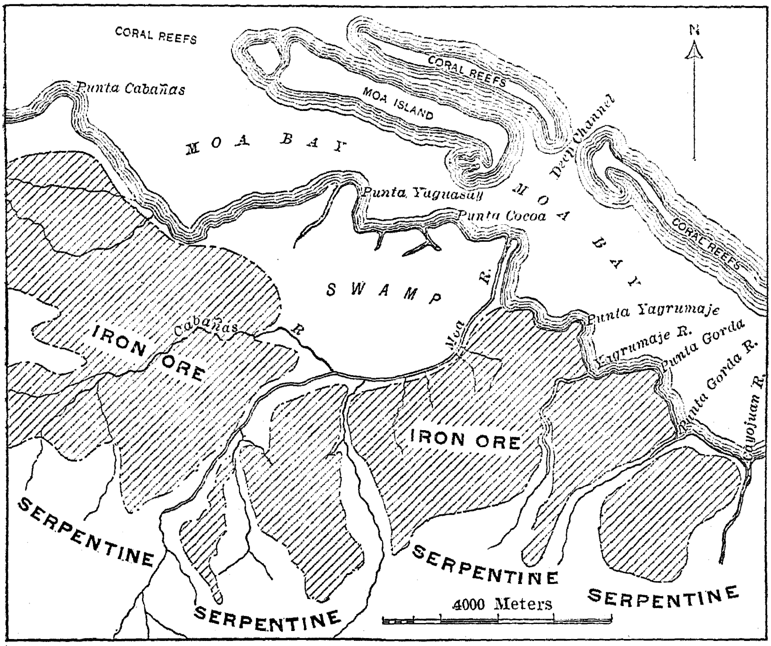
These analyses represent the entire mass of ore; that is, the clay-ore as well as the concretions and nodules. The figures are for the ores dried at 212° F. The ore in the ground frequently contains in addition up to as high as from 30 to 35 per cent. of hygroscopic moisture.

The general similarity of these ores is at once apparent, the

---

[4] *Iron Age*, vol. lxxx., No. 7, p. 424 (Aug. 15, 1907).

conspicuous features being low silica, high alumina, and the presence of chromium in very appreciable amounts. These features are more especially true of the ores from Mayari and Moa, and it should be noted that the analyses given for those fields are the averages of a much greater number of samples than for the Taco and Navas fields; hence they should be regarded as more truly representative. In addition to the ele-



The cross-hatched area indicates approximately that occupied by iron-ore. The western extremity of the deposit is not shown, but has been taken into account in estimating the tonnage of ore offered by the field.

FIG. 3.—SKETCH-MAP OF THE MOA IRON-ORE FIELD, SANTIAGO PROVINCE, CUBA.

ments given above, nickel has been found in amounts of from 0.5 up to, in some cases, 2.0 per cent. Manganese is also present. The balance of the ore comprises small quantities of magnesia and lime, with probably traces of the alkalies, and a very little sulphur.

Having established a common relationship between the ore-beds of these various localities, the discussion in its bearing upon questions of genesis will henceforth be more particularly

confined to the ores at Moa, since my opportunities for personal observation have been confined to that field. The Moa occurrence, moreover, as will appear later, approaches more nearly to a model type; at the same time, it will be seen that conclusions based on the Moa occurrence are equally applicable in their general aspects to the other occurrences. Before leaving the question of common relationship it may be well to review briefly the points from which this has been deduced, which are: (1) the blanket- or mantle-form of all the ore-beds; (2) the common bed-rock of serpentine; (3) the common appearance and nature—namely, ocherous clay-like materials carrying disseminated nodules and masses of brown ores; and (4) the common analysis, showing low silica and high alumina, with notable quantities of chromium and much combined water.

The map, Fig. 3, shows the general features of the Moa deposit. The ore-bed occupies practically the entire area adjacent to the shore, and extends thence inland for a distance of from 3 to 5 miles, finally fingering out along the crests of the divides. Wherever the ore comes to an end laterally, and where it has been cut through by streams, serpentine is found as the country-rock. A number of samples of this serpentine were taken, of which an average analysis is given in Table II., together with a reconstructed complete analysis of the ore.

TABLE II.—*Analyses of Country-Rock and Iron-Ore, Moa Bay, Cuba.*

	Serpentine. Per Cent.	Iron-Ore. Per Cent.
Fe_2O_3	......	66.90
FeO	8.55	......
SiO_2	37.29	1.71
Al_2O_3	1.33	11.60
TiO_2, Cr_2O_3, NiO	0.28[a]	
TiO_2		0.14
Cr_2O_3		2.65
NiO		0.60
P_2O_5	0.07	0.07
MnO	trace.	0.80
CaO	0.29	
MgO	36.53	
K_2O	trace.	
Na_2O	0.39	
CaO, MgO, K_2O, Na_2O		2.38[a]
H_2O	15.27	13.15
	100.00	100.00

[a] Calculated by difference.

The ores have already been described as "residual mantles," the inference being that they have been derived from the underlying rocks by processes of sub-aërial decay. Any other theory of origin involves transportation.

The theory that they are derived directly from the serpentine is supported by the analyses in Table II. The ore is, in fact, a laterite, a product due to the peculiar form of decomposition known as laterization, which is common to humid tropical climates. The essential characteristic of laterization is the breaking-up of the silicates, with the ultimate almost complete removal of the silica, wherein it differs radically from the kaolinization-processes of the temperate zones. Laterites have been reported from many tropical localities, and are especially common in India.

The ordinary procedure in rock-decay involves the removal of lime, magnesia, and the alkalies, while the aluminous silicates and the ferric oxides for the greater part remain behind. Laterization goes one step further and removes the silica as well. Its characteristics are: (1) the liberation of the silica from its various compounds; (2) the removal by solution of the lime and magnesia; (3) the oxidation of the ferrous to ferric iron; (4) the removal of the silica and the alkalies; (5) the concentration, as a residual mantle, of the alumina and ferric iron, with titania, chromic oxide, and other impurities; and (6) a sort of secondary dehydration leading to concretionary and pisolitic recemented masses, more or less abundantly disseminated through the mantle.

With this process in mind, the serpentine may be readily recognized as the parent of the iron-ore. Lime, magnesia, silica, and the alkalies have been largely if not wholly removed, and the iron and alumina have been concentrated. There is seven times as much iron in the ore as in the serpentine, and eight and one-half times as much alumina. About the same ratio appears to hold with the chromium, nickel, and titanium, which are nearly equally persistent with the iron and alumina. In short, there is no need to appeal to a hypothetical foreign source for any of the elements constituting the ore, either in whole or in part. No supposition involving transportation of material is required. Everything is at hand, and the history of the ore, as residual material derived directly from its underlying rock, is complete.

Reference has been made to the laterites of India. These laterites, being derived out of a different parent-rock from that in Cuba, naturally result in different end-products. Two types are recognized in India: the "high level" and the "low level." The former are *in situ*, while the latter are detrital, and differ from the usual detrital rocks only in the amount of lateritic cement. They have furnished the supply of iron-ore for numerous small native iron-smelting centers, and in this character they suggest an analogy with the Moa ores. As a matter of fact, however, they are transported materials, and their iron-content, as found to-day, is undoubtedly due to mechanical sorting and concentrating, as well as, very possibly, to a certain amount of secondary chemical concentration. The presence of large amounts of mechanically-intermingled quartz still further distinguishes this type of lateritic ores from those at Moa. The true analogy lies between the Moa iron-ores and the Indian "high level" laterites, or (as they have within recent years in many cases proved to be) bauxites. The bauxites are found in the form of residual mantles, overlying their parent-rocks in a manner exactly similar to the occurrence of the Moa ores. The parent-rock, however, is dolerite and not serpentine. The result of laterization is therefore an end-product consisting chiefly of alumina, with ferric iron and other persistent impurities in proportion to the content of the original rock. Secondary dehydration marks the final stage of the process, as with the Moa ores, the result being a recemented pisolitic and nodular, frequently vesicular, reddish-brown material, resembling very closely in its appearance the recemented brown ores of Moa and other Cuban deposits. In short, whereas the parent and the offspring in the two cases respectively differ from one another, the process of generation is undoubtedly closely similar.

Dr. H. Warth[5] has presented a very interesting pair of analyses, one of dolerite and the second of bauxite, directly derived therefrom; both samples are from the western Ghats, near Bombay, India. These analyses are reproduced in Table III. for the purpose of comparison with the analyses of the Moa serpentine and iron-ore, given in Table II.

[5] *Geological Magazine*, N. S., decade V., vol. ii., No. 1, p. 21 (Jan., 1905).

TABLE III.—*Analyses of Dolerite and Bauxite, India.*

	Dolerite. Per Cent.	Bauxite. Per Cent.
SiO_2	50.4	0.7
TiO_2	0.9	0.4
Al_2O_3	22.2	50.5
Fe_2O_3	9.9	23.4
FeO	3.6	......
MgO	1.5	......
CaO	8.4	......
K_2O	1.8	......
Na_2O	0.9	......
H_2O	0.9	25.0
	100.5	100.0

The disappearance of the silica, lime, magnesia, and the alkalies, and the concentration of the alumina and iron, are here beautifully exemplified. There is 2.25 times as much alumina and 1.70 times as much iron in the bauxite as in the dolerite; thus a part of the iron has been removed, as was the case with the Moa ore. The titania also appears to have suffered partial removal. The solution of the silica is wonderfully complete.

Enough has been said to demonstrate that the Moa type of Cuban ore has its genesis through sub-aërial decay directly out of the underlying rock. The decay takes that form which has come to be known as laterization, a form peculiar to humid tropical climates, which has as its essential characteristic the breaking-up of the silica compounds and the more or less complete removal of the silica. The process has come nearest to completion at Moa, and that name may, therefore, be properly adopted to designate the type. The ores of other localities, Mayari, Cubitas, Taco, and Navas, are merely less-perfect examples of the same process.

The strange fact that laterization is confined to humid tropical climates has been the subject of more or less speculation. Sir Thomas Holland, Director of the Geological Survey of India, in his paper, On the Constitution, Origin, and Dehydration of Laterite,[6] has proposed a very ingenious and interesting theory in this connection. It should be remembered that he is speaking of lateritic bauxites which have been derived from dolerites, and, therefore, the reference is to aluminous silicates. It

---

[6] *Geological Magazine*, N. S., decade IV., vol. x., No. 2, p. 59 (Feb., 1903).

does not seem improper, however, to read magnesian for aluminous; in other words, the theory appears as applicable to the breaking-down of serpentine as of dolerite.

Sir Thomas Holland says:

"To account for the fact that an aluminous silicate undergoes a more complete disintegration under tropical conditions than under the deep-seated and presumably high temperature conditions of kaolinization, the writer suggests that laterite is due to the agency of lowly organisms, possibly akin to the so-called nitrifying bacteria. With these are probably forms akin to the bacteria which oxidize and fix ferrous compounds, and which, precipitating the silica in the colloid form, permit its removal by the dilute alkaline solutions simultaneously formed."

We may now turn from the question of genesis to a consideration of the probable age and history of the Moa type of brown-ore deposits. In this connection, I wish to acknowledge my indebtedness to A. C. Spencer for suggestions taken not only from his publication but also from verbal discussions.

The Cubitas, Mayari, Navas, and Taco deposits all occupy plateaus. Mr. Spencer says of the Cubitas in his paper, Three Deposits of Iron Ore in Cuba:[7]

"Within an area measuring roughly 10 miles east and west and 4 miles north and south, there are several flat-topped mesas rising 300 to 400 feet above the general level of an almost featureless plain. . . . The ore deposits are all surface mantles covering the plateau-like mesas."

The actual elevation of the *mesas* above sea-level is not given. The following description of the Mayari deposit has also been published:[8]

"The ore body is on the summit of a gently rolling plateau, roughly 10 miles long and 4 miles wide, with its principal axis lying northeast and southwest. Its elevation is about 1600 ft. at the northwestern extremity, which is nearest to Nipe Bay, and it rises toward the southwest to an elevation of 2200 to 2300 ft., with one peak reaching to 2600 ft. and another to 3200 ft.

The Taco deposit occupies a partly-dissected plateau, approximately 4 miles long by 1.5 miles wide, with the longer axis NE–SW., at a general elevation of 2,100 ft. above sea-level. The Navas plateau, extending 2.5 miles N–S. and 1.25 miles wide, is from 1,600 to 1,800 ft. above sea-level.

At Moa the conditions are different. The ore-mantle, adjacent to tide-water and extending thence inland for a distance of

[7] *Bulletin No.* 340, *U. S. Geological Survey*, p. 324 (1908).

[8] *Iron Age*, vol. lxxx., No. 7, p. 421 (Aug. 15, 1907).

from 3 to 5 miles, lies on a surface exhibiting no very pronounced relief, the average grade shorewards being about 250 ft. to the mile. The final disappearance of the ore-mantle inland is obviously due to the usual processes of surface-denudation. The underlying serpentine first appears in the stream-beds, which may be wholly in bed-rock, with the ore still persisting on the inter-stream divides. Thus, the upper edge of the ore-body presents a series of fingers persisting upwards along the crests of the divides till it finally disappears on the divides as well as in the stream-bottoms.

There can be little doubt that the Moa type of ores were originally formed on an ancient peneplain, and probably occupied at that time vastly greater areas than to-day. The present plateaus are remnants of the ancient peneplain, and owe their elevated position to a period of uplift, accompanied by a certain amount of warping, fracturing, and probably partial subsidence. The plateau-deposits underwent more or less simple vertical uplift. The Moa area was tilted and possibly fractured, subsiding once more to somewhere near its present position and level. The peneplain period may perhaps be referred to the Upper Oligocene, since at that time, according to Hayes, Vaughan, and Spencer,[9] nearly the whole island of Cuba was submerged, excepting portions along the north and south shores of Santiago Province. The next succeeding (Miocene) period was one of general uplift. Quoting from the report of the above:

> "There was folding and uplift during this period, the elevation along the axial line being greater than at the sides. . . . One would infer that the central portion of the Province of Santiago was more highly elevated than the coastal portion, since Upper Oligocene limestones occur in the central portion of that province at considerably higher elevations than along either the north or the south coast."

This last statement probably explains the tilting of the Mayari plateau towards the north, and also the tilting and possibly the breaking-off and resulting partial subsidence of the Moa block.

The uplift was followed by a rejuvenation of the streams and renewed denudation activity. The ancient peneplain was dissected and cut back until there were left only the present rem-

[9] *Report on a Geological Reconnoissance of Cuba*, included in *Civil Report of Brig.-Gen. Leonard Wood, Military Governor of Cuba*, vol. i., pp. 31 to 34 (1901).

nants, still carrying their original mantles of ore; and even these are, in a geological sense, still rapidly disappearing. The upper portion of the Moa block has been planed off, while the lower portion owes its preservation to the fact that it lies so near sea-level that the stream-gradients have practically disappeared, and their power to dissect and remove has been reduced to a minimum.

Thus, the original development of the Moa type of ores should probably be referred to pre-Miocene times. During and since the period of uplift, vast quantities of these ores have undoubtedly been mechanically removed and dissipated. At the same time there is no reason to suppose that laterization-processes have ceased; it is in fact probable that new ores are forming to-day wherever opportunity offers. Such opportunity may be regarded as at a minimum on the plateaus, where heavy mantles of material lying nearly horizontally effectively protect the underlying rock from the action of surface-waters. Wherever the stream-beds have dissected the plateaus, however, revealing fresh areas of serpentine and inducing new and lower curves in the adjacent ground-water levels, the growth of laterite must be proceeding as it did in former times. The presence of numerous sinks appears to furnish evidence to support this statement. Thus, the descent of the lateritic zone must be keeping pace with the degradation of the land-surface, except where this is so rapid that products of decay are removed as soon as formed.

We infer, then, that there are two limits to the accumulation of laterite. On the one hand, too great an opportunity, which has led to its development on an undisturbed peneplain up to a point where the underlying rock is fairly well protected—in this case the decay will no doubt continue to proceed downwards, but at a greatly reduced rate; and, on the other hand, too little opportunity, through its removal mechanically as fast as it forms. The first limit may have been the condition of affairs at the close of the Upper Oligocene, when huge mantles of laterite had accumulated, to about their maximum depth. The second limit is that of to-day, except where, locally, intermediate sets of conditions may exist. At such points the ore-beds are undoubtedly increasing. On the whole, however, since the beginning of Miocene times, the probabilities are that very much greater

quantities of the lateritic ores have been destroyed than have been formed.

The Moa deposit differs from the others chemically in being a more perfect type of laterization, and, structurally, in not occupying a plateau. It would appear that the deposit was originally a portion of a great peneplain, upon which was accumulated a thick mantle of ore through processes of sub-aërial decay common to humid tropical climates. At the time of the general uplift the Moa block apparently broke off and partly subsided once more, tilting toward the north. Renewed denudation then planed off the upper edges of the block, removing the ore-mantle and cutting down into the underlying rock. Any further ore forming at the higher levels was removed as fast as it formed. Lower down, however, transportation was at its minimum, and the ore already existing not only remained undisturbed, but probably increased somewhat in depth. Furthermore, through saturation by surface-waters, the process of laterization was advanced nearly to theoretical completeness, and at the lower levels, representing the heart of this great ore-field, there are occasionally found mantles of ore exceeding 60 ft. in thickness, containing a minimum of silica.

The structural history of this field may raise the question, why attribute the accumulations of ore entirely to decay in place? Was there not exceptional opportunity here for mechanical transport, or transport in solution with secondary concentration? The answer is, the absolute absence of evidence or appearance of such accumulation through transport of material. The ores as found are satisfactorily accounted for in their entirety by the decay of the underlying rock. The analyses of many samples, representing in a most thorough manner wide areas, are remarkably uniform, although many were taken from localities where opportunity for secondary concentration or accumulation would be much better than at other localities. In the absence of any evidence to the contrary, the conclusion must be that the ore-bearing materials which have been removed from the upper levels were simply dissipated by the waters removing them. They do not appear to have contributed to the ores of the lower levels, either by increasing the bulk of the accumulation or by any process of enrichment.

Similar questions of possible transport of material cannot be

raised by the structural conditions attendant upon the other fields. Hence, the theory of development of ore in place through laterization-processes offers the only ready and entirely satisfactory explanation for the ore-deposits in those fields; and that the laterization has not been as perfect as at Moa may be due to any number of readily-suggested causes.

The development of these huge fields of iron-ore in Cuba has directed study upon several metallurgical problems attending their use in the manufacture of iron and steel, chiefly in connection with the high alumina- and chromium-content. Exhaustive studies and experiments on Mayari ores have been carried out within the last few years by the Pennsylvania Steel Co., and it has been announced that all the difficulties have been solved, and steel rails of more than usual excellence have been manufactured from these ores. It is not within the scope of this paper to go into metallurgical details. I merely refer to this feature and its happy solution in a congratulatory frame of mind. The cheapness with which the ore can be extracted must have presented itself to any one of mining experience. The high moisture-content, hygroscopic plus combined, together with the clayey structure, makes it very desirable to treat the ore before shipping by some agglomerating-process under fairly-high temperatures. With the former difficulties attending the presence of high alumina and high chromium removed, the agglomerated or nodulized ore, practically free from moisture and in that condition carrying more than 50 per cent. of metallic iron, will undoubtedly find a ready market on the Atlantic seaboard, and in all probability new plants will be erected on the seaboard to profit by this opportunity.

Monazite and Monazite-Mining in the Carolinas.

BY JOSEPH HYDE PRATT, CHAPEL HILL, N. C., AND DOUGLAS B. STERRETT, WASHINGTON, D. C.

(Chattanooga Meeting, October, 1908.)

I. Description.

Monazite is one of the minerals which, for a long time, was considered somewhat rare in its occurrence, but, upon a commercial demand arising for it, prospectors and engineers soon located large deposits of it in the Carolinas and Brazil, and the supply has always been able to meet the demand. During 1908 further sources of supply of monazite have been discovered and developed in Idaho. North and South Carolina, however, are the only States that have thus far put any monazite on the market.

Monazite is essentially an anhydrous phosphate of the rare-earth metals, cerium, lanthanum, and didymium, $(Ce,La,Di)PO_4$. There is nearly always present a varying but small percentage of thoria, ThO_2, and silicic acid, SiO_2, which are very probably united in the form of a thorium silicate, $ThSiO_4$. Some monazites contain but a fraction of a per cent. of thoria, while others have been recorded that showed the presence of from 18 to 32 per cent.; but the majority contain from 3 to 9 per cent. of this oxide. It is the presence of the thorium oxide that gives the monazite its commercial value. The analysis occasionally shows also the presence of other constituents, as the yttrium and erbium oxides, zirconia, alumina, magnesia, lime, iron oxides, manganese oxide, and titanium oxide.

Monazite is light-yellow, honey-yellow, reddish-, brownish-, or greenish-yellow in color, with a resinous to vitreous luster, and is translucent to subtransparent. It is brittle, with a conchoidal to uneven fracture, and is from 5 to 5.5 in hardness, and from 4.64 to 5.3 in specific gravity. It crystallizes in the monoclinic system, and some crystals have been observed that were 6 in. in length. The more perfect crystals are, however,

very small, ranging in length from 0.5 in. down to microscopic sizes. The mineral is usually readily recognized after a few samples have been examined. The principal chemical and blowpipe reactions that can be readily employed to identify monazite are the following: It is incompletely soluble in hydrochloric acid, but is completely and readily acted upon by sulphuric acid. If oxalic acid is added to the very dilute, filtered, sulphuric acid solution, or to the acid solution obtained by fusing the mineral with soda, a precipitate is obtained which, upon ignition, becomes brick-red, due to cerium oxide. Before the blow-pipe the mineral turns gray, but is infusible. If heated with sulphuric acid, it colors the flame bluish-green, due to phosphoric acid.

The thoria-content of the monazite, the substance for which the mineral is mined, varies from 0.01 to more than 7 per cent. The following analyses illustrate the variation in the percentage of thoria present:

TABLE I.—*Percentage of Thoria* (ThO_2) *in Monazite Sand of North Carolina.*[1]

	ThO_2. Per Cent.
White Bank gold-mine, Burke county,	2.15
Hall's Creek, Burke county,	2.25
Linebacher place, Silver Creek, Burke county,	6.54
Long Branch, McDowell county,	1.27
Alexander Branch, McDowell county,	6.30
Mac Lewrath Branch, McDowell county,	2.48
Proctor farm, near Bellwood, Cleveland county,	5.87
Wade McCurd farm, Carpenter's Knob, Cleveland county,	6.26
Davis mine, near Mooresboro, Cleveland county,	3.98
Henrietta, Rutherford county,	1.93

The results in Table I. are for the concentrated sand, but in a number of cases the sand could have been concentrated to a higher degree of purity, and a higher percentage of thoria thus obtained.

II. OCCURRENCE.

1. *Geography.*—Monazite is of wide-spread occurrence in the United States, though commercial deposits have been found in but few regions. The area in which monazite-deposits of commercial value have been found in the Carolinas lies in the south

[1] *Bulletin No. 9, North Carolina Geological Survey,* p. 21 (1895).

central part of western North Carolina and in the extreme northwestern part of South Carolina, as indicated in the sketch-map, Fig. 1.[2] This area covers about 3,500 sq. miles, and includes part or all of Alexander, Iredell, Caldwell, Catawba, Burke, McDowell, Gaston, Lincoln, Cleveland, Rutherford, and Polk counties in North Carolina; and Cherokee, Laurens, Spartanburg, Greenville, Pickens, Anderson, and Oconee counties in South Carolina. The larger towns within or near the monazite-region in North Carolina are Statesville, Hickory, and Shelby; and in South Carolina, Gaffney, Spartanburg, and Greenville. This monazite-region is crossed by the Southern,

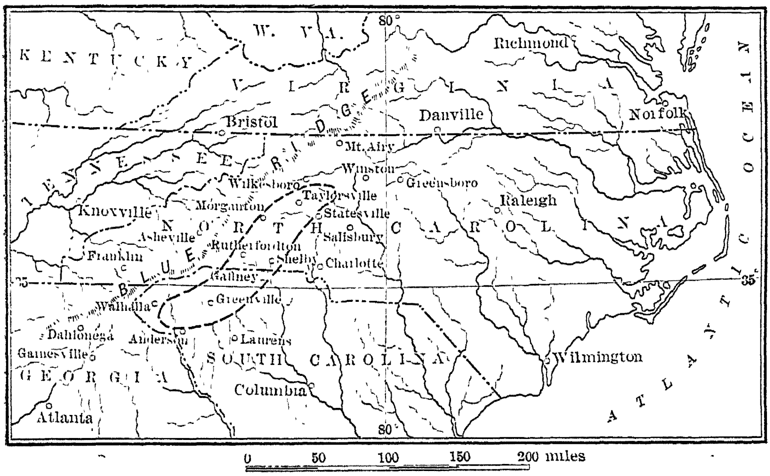


FIG. 1.—MAP SHOWING AREA OF THE CAROLINA MONAZITE-REGION.

the Seaboard Air Line, and the Carolina & North Western railroads.

Several deposits of monazite have been located in northeastern Georgia, though their value has not yet been determined. One of these in Rabun county showed a good quantity of both gold and monazite in a preliminary test. In the adjoining Jackson county of North Carolina, monazite was found in several pannings that were made in the Horse Cove region, two miles east of Highlands. At a number of other places in the mountain-region of North Carolina monazite occurs in pegmatized gneisses and schists. Several small deposits of fairly-rich

[2] Reproduced from *Bulletin No.* 340, U. S. Geological Survey, p. 274 (1908).

monazite-bearing gravels are reported by George L. English to occur in Clay county, N. C. The lack of large areas of bottom-lands, however, limits the value of these deposits. It has also been found to a limited extent in Cub creek, near Wilkesboro, Wilkes county, N. C.

2. *Physiography.*—Physiographically, North and South Carolina are divided into three parts. These are the Coastal Plain, extending from the Atlantic ocean NW. for from 100 to 150 miles; the Piedmont Plateau, extending from the limits of the Coastal Plain NW. for from 100 to 130 miles to the foot of the Blue Ridge; and the mountain-region, extending NW. from the Piedmont Plateau to the State lines. The Coastal Plain and the Piedmont Plateau are prominent in both States, but only North Carolina contains a large portion of the mountain-area.

The Coastal Plain is a broad, nearly flat stretch of country rising from sea-level on the southeast to an elevation of a few hundred feet on the northwest, in which direction it is practically limited by the boundaries of the rock-formations of which it is composed. The Piedmont Plateau is an elevated district rising from a few hundred feet above sea-level on the southeast to from 1,200 or 1,500 ft. on the northwest. It forms a plateau much dissected by valleys from 50 to 200 or 300 ft. deep, and its regularity is further disturbed by scattered mountain-peaks and smaller hills rising above its general level. The features of the plateau are best observed from a prominent ridge or one of the smaller hills of the region. In the mountain-region are included the Blue Ridge and its foot-hills, and the higher mountains to the northwest. The country in the mountain-region is exceedingly rough, and the elevations range from 1,500 to more than 6,500 feet.

The region in which valuable deposits of monazite have been found may be defined as a belt from 20 to 30 miles wide and more than 150 miles long, lying wholly within the Piedmont Plateau and bordering closely on the Blue Ridge, to whose general course it is roughly parallel.

III. Geology.

1. *Formations.*—The rocks of the Carolina monazite-region are principally gneisses and schists. These include the Caro-

lina and Roan gneisses,[3] granite-gneiss, and porphyritic granite-gneiss. Among other rocks are massive granite, pegmatite, peridotite and allied rocks, quartz-diorite, and diabase.

The Carolina gneiss is of Archæan age and is the oldest and most important rock of the region. It is composed of several types of gneisses and schists which exhibit various degrees of metamorphism. The most common types are mica-, garnet-, cyanite-, and graphite-gneisses and schists, or combinations of two or more of these types. The mica of the micaceous types may be either biotite or muscovite, or both. More or less mica is generally present in all of the types of the Carolina gneiss, while the garnet- and cyanite-types, with or without the graphite-type, also occur together. The different types of the Carolina gneiss vary in color from light gray to dark gray and are sometimes bluish-gray or bluish-black where graphite is abundant in them. Some types of the Carolina gneiss are fine-grained, so that the component minerals are distinguished with difficulty, while others are more coarsely crystallized. Some of the common constituent minerals of the Carolina gneiss are biotite, muscovite, quartz, garnet, cyanite, feldspar, and graphite. The presence of much pegmatitic material is a characteristic feature of much of the Carolina gneiss.

The Roan gneiss is the next oldest formation of the monazite-region and is also of Archæan age. It consists of hornblende-gneiss and schist, with occasionally the less-metamorphosed phase, diorite. The hornblende-gneisses and schists are composed chiefly of small interwoven and matted hornblende crystals, and grade into diorite, which contains a noticeable amount of feldspar and has a granitoid texture. The hornblende-rocks vary from black to dark green in color. Bands of mica, gneiss, and schist, possibly of the Carolina gneiss, are included in both large and small masses of the Roan gneiss.

The age of many of the granites and granite-gneisses has not been determined, though a part are probably Archæan. The granites and their different phases are next to the Carolina gneiss in importance, and are particularly prominent in areas where rich deposits of monazite exist. The types found in the monazite-region are biotite-granite, muscovite-granite, and horn-

[3] Formation-names are taken from *Folios Nos.* 90, 124, 143, 147, *and* 151, *U. S. Geological Survey*, by Arthur Keith.

blende-granite, while in some places considerable secondary garnet has developed in the gneissoid granites. The textures of the granites are gneissic or schistose, porphyritic, and massive. Where the granite is both porphyritic and schistose the feldspar phenocrysts often have an augen form, caused by crushing and shearing. Many of the granite masses have much quartz in veins and veinlets throughout their mass. Some of this quartz is massive crystalline, and some occurs with more or less well-defined crystal form, or drusy surfaces. The occurrence of quartz-veins is not always confined to the granite masses, but in many places extends some distance from the contact of the granite into adjacent formations. The composition of the granite masses near the contact with other formations has in many cases been altered by the part or complete absorption of inclusions of those formations. This phenomenon is particularly evident where a mica-granite, by intrusion into a mass of Roan gneiss, has become a hornblende-granite near its borders through the absorption of hornblende.

Pegmatite is a common rock throughout the monazite-region, and is especially prominent in those areas rich in monazite. Two principal methods of occurrence are here recognized. In one, the pegmatite occurs in distinct masses or bodies composed of quartz and feldspar, with or without mica and other accessory constituents. The texture of these masses is, in some cases, extremely coarse, with the minerals composing the pegmatite separated out in crystals or masses many inches across. The other type is pegmatized gneiss, representing the addition of the pegmatite minerals to the gneiss, with perhaps some re-crystallization of portions of the inclosing rocks. The nature of this pegmatized rock varies considerably. In some places secondary quartz is the principal mineral added, while feldspar is present in smaller quantities. In others feldspar is more prominent. Mica may or may not be present in the pegmatitic material, but has generally been plentifully developed in the mass of the gneiss by metamorphism. The feldspar of pegmatized gneiss often assumes a porphyritic form, producing augen-gneisses. The gneisses and schists are often banded with or cut at all angles by streaks of pegmatitic or granitic material. The recrystallization of the gneisses and schists, with the development of pegmatitic material or the injection

of such material through the rocks, may be called pegmatization. In many places the process has proceeded so far that it is very difficult to distinguish pegmatized gneiss from granite-gneiss, especially from porphyritic and flow-banded granite-gneiss. This difficulty is partly due to the fact that granite and pegmatite are composed of the same minerals and have no sharp division-line between the size of their grains.

The peridotites and allied basic rocks are dark green to greenish-black in color, and contain one or more of the ferro-magnesian minerals, olivine, pyroxene, and hornblende, as chief constituents. So far as known, these rocks are of Archæan age, and are probably genetically connected with the Roan gneiss. Though a relatively unimportant rock of the monazite-region, these basic rocks generally outcrop prominently wherever they occur, and many of the outcrops are marked by large rounded "nigger-head" boulders. The peridotites and allied rocks are often altered to talcose or chloritic soapstone or serpentine. In some cases this alteration is only superficial, but in others whole masses have been so metamorphosed. These rocks generally occur in lens-shaped bodies parallel, or nearly so, to the schistosity of the inclosing rocks.

Quartz-diorite of undetermined age is one of the less important intrusive rocks of the monazite-region. It is a hard, fine-grained rock, composed of granular quartz and feldspar with varying quantities of hornblende. Locally, garnet is distributed promiscuously through it. Quartz-diorite occurs in small dikes, from a few inches to several feet thick, cutting the formations at various angles. Their size is off-set by their abundance in some sections and their resistance to erosion, owing to which they leave much débris over their outcrops in the form of hard, rounded boulders.

Diabase, probably of Triassic age, is the latest intrusive rock known in the monazite-region. It is a dense, hard rock of dark-green to black color, composed chiefly of olivine and a lime-feldspar, and is rather abundant in some sections, occurring in dikes from a few feet to more than 100 ft. wide. The outcrop is generally marked by abundant characteristic spheroidal "nigger-head" boulders. The diabase-dikes cut the rocks at various angles, though in many cases they have a north to northwest strike.

2. *Structure.*—The rocks of this region have undergone extreme regional metamorphism, with accompanying folding and faulting. The mashing and recrystallization of the rocks of the Carolina gneiss-formation have been so extensive, in some cases, that much of the original sedimentary structure and igneous texture have been destroyed. The folding of the older formations has resulted, in some places, in complex structures of both large and small dimensions. Some of the folds extend over miles of region, while others are confined to a few feet or inches. The minor deformations and crumplings—miniature Appalachian folds—seen in some rock-exposures portray the form of the larger folds. The Carolina gneiss has been intruded by rocks of later age and cut by them into irregular-shaped masses, many of which fork out into long tongues or occur as narrow streaks in the intrusives, or *vice versâ*. There have been successive intrusions of igneous rocks of later age into the earlier formations. Thus, the Carolina gneiss is cut by the Roan gneiss, and both are cut by granites of later age.

The structure of the pegmatite in this region is quite variable. In some places the pegmatite occurs in sheets or lenses, interbedded and folded with the inclosing gneisses and schists. In other places it occurs in dikes, veins, or lenses, either conformable with the inclosing rocks through part of its extent, and cutting across them in other parts, or in irregular masses having no definite orientation in the surrounding formations. In pegmatized rock-masses pegmatization has generally affected certain beds, which grade into regular pegmatite in the direction of either their greatest or their least extension. In such rocks it is often impossible to determine the line of demarcation between the two. There is also a gradation between the pegmatized beds and ordinary gneiss.

3. *Rocks and Soils.*—The rocks of the southern Appalachian region have undergone extensive weathering, and in many places, in the Piedmont Plateau especially, are concealed by a thick mantle of residual soil. In many sections good outcrops are scarce, and are found mostly on steep hill-sides, along water-courses, and in road-cuts. The residual soils often furnish evidence of the nature of underlying rocks, and can be used as a guide to their determination. It is first necessary

to learn the different stages of soil-formation by the examination of many outcrops and their gradations into residual soil.

The Carolina gneiss, on part disintegration and decomposition, commonly forms a gravelly soil, with a red clayey matrix. This is especially characteristic of the garnetiferous and graphite-cyanite types, which are abundant in parts of the monazite-region. The pebbles are composed of small fragments of the original rock, such as tufts of cyanite impregnated with hematite or limonite, iron-stained garnets, or pieces of hematite. On more complete decomposition, a fine reddish clayey soil results, with no decided characteristics. Other types of the Carolina gneiss, in which mica is an important constituent, leave a micaceous soil, much of which assumes a purplish color. Granite and its various phases, on part disintegration and decomposition, yield light sandy soils. On more complete decomposition, the granites yield soils of a light- to dark-reddish color, depending on the quantity of ferro-magnesian minerals, as biotite or hornblende, in the original rock. The quartz-grains of the granite remain as sand mixed through a clayey matrix. This quartz sand is almost everywhere to be seen at the immediate surface, from which the clays have been washed by rains. Where Carolina gneiss and granite are intimately associated, or where pegmatization has been extensive in a body of Carolina gneiss, there results a sandy soil, characteristic of granite, through which are scattered pebbles of hematite and ferruginous cyanite, characteristic of the Carolina gneiss. The relative importance of pebbles in such soils decreases as the quantity of pegmatite or of granite in the rock-formations increases. These features of the soils are especially marked on the broad, flat ridges characterizing much of the Piedmont Plateau region. The Roan gneiss leaves a greenish sandy soil on disintegration, and an ocher-yellow to dark reddish-brown or chocolate-colored clayey soil on decomposition. Black stains of manganese are associated with many of the soils derived from hornblendic rocks.

A clew to the nature of the rock-formations in a given region is often furnished by the character of the gravels in the bottom-lands and streams draining that region. Thus, in this area a very light-colored gravel, with much quartz débris, indi-

cates a granite, or its contact, or a very highly pegmatized country-rock. Garnets and hematite iron-ore, with which blocks of mica- or cyanite-gneiss are associated, indicate Carolina gneiss. Quantities of black sands in the stream-gravels, containing magnetite, ilmenite, hornblende, etc., are characteristic of the Roan gneiss.

Monazite has been found in several varieties of rocks, in the soils derived from monazite-bearing formations, and in gravel-beds formed through the erosion of these formations. Only gravel-deposits have been profitably worked for monazite on an extensive scale, though in some places the surface-soils adjoining rich deposits of monazite, or the saprolite or rotted rock underlying them, are found to be sufficiently rich in monazite to be sluiced down and washed.

The percentage of monazite in both the original rock-matrix and the gravel-deposits is small, and probably does not often exceed 1 per cent. Figures are not available for the percentage of monazite in gravel-deposits. From the saprolite underlying the F. K. McClurd mine, 0.75 mile NE. of Carpenter Knob, N. C., George L. English obtained about 5 oz. of monazite per ton, or about 0.016 per cent. At the British monazite-mine, 3 miles NE. of Shelby, N. C., the quantity of monazite in the hard-rock formations was found by Hugh Stewart, engineer in charge, to run from less than 0.03 per cent. up to more than 1.1 per cent.

Monazite has been observed in the Carolinas in several types of rock, among which are gneiss, pegmatized gneiss and schist, pegmatite, and different varieties of granite. The occurrence of monazite in ordinary pegmatite masses is in large masses or crystals. These have been found varying from an ounce or two to 60 lb. in weight in the pegmatites of Mitchell and Madison counties, N. C.

Most of the pegmatized-gneiss bodies, which are rich in monazite, represent phases of the Carolina gneiss, in which the original nature of the rock has been largely obliterated as a result of the addition of new minerals and the recrystallization of the original ones into pegmatitic material. The texture developed during this pegmatization is in many cases porphyritic, in which the feldspar phenocrysts assume somewhat of an augen form. The feldspar phenocrysts range in size from some

smaller than a grain of wheat to others the size of a walnut. The porphyritic gneiss may grade into less or more highly pegmatized gneiss, and from the latter into regular pegmatite. This gradation may be between two separate beds or from one part to another of the same bed. In those beds or portions of beds where there has been little pegmatization monazite occurs sparingly. The same is true where pegmatization has been complete, and but little of the original gneiss remains. It is, then, the beds of gneissic rock which are rich in secondary quartz, and contain numerous small masses of feldspar throughout, that carry the most monazite. In such rocks there is generally much biotite, with graphite and perhaps some muscovite and other accessory minerals, as well as abundant quartz and feldspar. The quartz occurs in layers or scattered grains throughout the rock, inclosing and replacing the other constituents. The feldspar crystals chiefly replace, though they partly displace, the other minerals of the rock. Monazite in a rock matrix almost invariably possesses crystal form, often with brilliant faces.

As a typical example of rich monazite-bearing rock, that from the mine of the British Monazite Co., 3 miles NE. of Shelby, Cleveland county, N. C., is chosen for description. Fig. 2 represents a section across a hand-specimen of this rock, and shows the main features to which attention will be called.[4] The chief constituents of this rock are quartz, feldspar (mostly the potash variety), biotite, graphite, muscovite, monazite, and a little zircon. The specimen has a banded structure, caused by the more or less separate occurrence of certain minerals arranged in parallel streaks, with a roughly-parallel orientation of the crystals or grains of each mineral. The principal features of the banding, as seen in the section, consist of one large quartz streak, with several smaller streaks and individual grains, in a regular biotite-schist. The other minerals of the section occupy various positions, and show diverse relations to the minerals of these bands and to each other. The feldspar is porphyritic and occurs chiefly in individual crystals, some of which are of considerable size. A number of the feldspar phenocrysts are small bodies of pegmatite in themselves. As an example, the largest feldspar crystal shown in the section includes

[4] Reproduced from *Bulletin No.* 340, *U. S. Geological Survey*, p. 282 (1908).

both quartz and muscovite. The feldspar at the lower left-hand side of this crystal also has much quartz and muscovite associated with it. As shown in the section, the feldspar phenocrysts replace the other minerals. This replacement is especially well shown by the interruption, with but little displacement, of the lower biotite band by the large crystal described above. Graphite occurs in large amounts with biotite, though it is associated with nearly every other mineral of the rock. Where present, muscovite is chiefly associated with the feldspar. Monazite seems to be indiscriminately scattered through the rock, included in or associated with all the foregoing min-

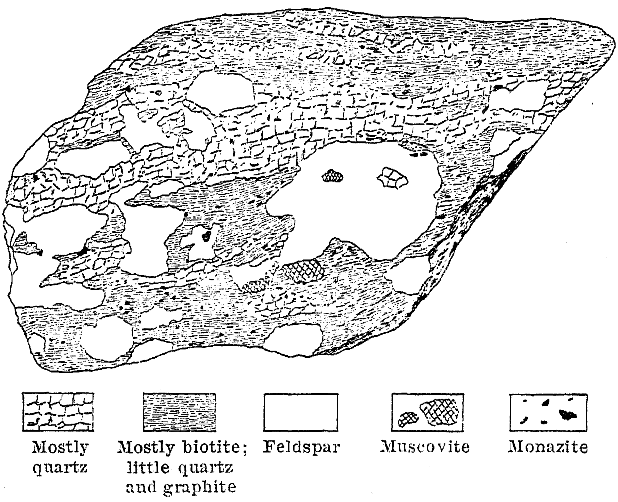


FIG. 2.—SPECIMEN OF MONAZITE-BEARING ROCK (THREE-FOURTHS NATURAL SIZE), FROM BRITISH MONAZITE CO.'S LAND, 3 MILES NORTHEAST OF SHELBY, N. C.

erals. Though generally free from inclusions, it is not invariably so, and in one case a plate of graphite was observed within a monazite crystal. All the minerals observed in the rock, with the exception of zircon, have been noted as inclusions in the feldspar phenocrysts.

In microscopic sections cut from specimens from one of the ore-streaks the minerals described above were observed, together with some iron-staining. The feldspar is principally orthoclase and microcline, partly kaolinized. The quartz is plainly secondary, and occurs in bands or streaks or grains parallel with the schistosity of the rock. In some places the

quartz has been deposited in the fractures or between the grains of other minerals; in others, it replaces or includes fragments of such minerals as biotite and graphite.

Gas-cavities and inclusions of very fine acicular needles, probably rutile, are abundant in the quartz. Biotite occurs in interwoven laths and crystals roughly parallel to the banding of the rock. The pleochroism of the biotite is light yellow-brown to greenish-brown or dark purplish-red. Graphite occurs as plates and laths, in general lying parallel to the banding of the rock. Some of it is interbanded and even interleaved with biotite; elsewhere the plates are turned across the foliation. In one section a lath of graphite was observed inclosed in quartz which filled a fracture across the foliation of a biotite crystal. Monazite occurs in contact with the various minerals of the sections, though it is more commonly surrounded by or included in grains of biotite and quartz. The position of the monazite in the biotite indicates replacement, and the biotite foliæ are not displaced around the crystals. In the microscopic sections the feldspar observed was not sufficient to determine its relation to the other minerals.

The rock has been so thoroughly recrystallized that it is difficult to give the relative order of formation of the minerals. Biotite, if not still in its original condition, was probably the first mineral to form during recrystallization. Part of the graphite was probably contemporaneous with the biotite. Some, however, was introduced later and formed at the same time with the quartz. The small amount of muscovite in the rock was probably next to form, followed closely by quartz. From the small amount of feldspar in the microscopic sections, it was not possible to state its relative period of formation. From the hand-specimen, however, shown in Fig. 2, it is evident that the feldspar was introduced later than the quartz, or possibly contemporaneously with part of it. An attempt was made to mine the monazite-rock, but the percentage was too small to make mining profitable, although it was possible to obtain a very clean monazite by separation and concentration.

4. *Origin.*—The occurrence of monazite in granitic and pegmatitic rocks indicates that its origin is associated with magmatic agencies. It is probable that the constituents of monazite are associated with granitic magmas, and that only part of

the mineral crystallizes out when such magmas solidify. During the formation of pegmatite magmas and solutions from the residues of the solidification of granite, part of the constituents of monazite are retained. When these pegmatite magmas and solutions are intruded into or deposited in the gneisses and schists in masses such as are mined for mica, the monazite forms in large masses or crystals. During the pegmatization of rock-formations by these magmas and solutions the monazite is carried into the gneisses and schists, where it is now found. This pegmatization with which monazite is associated was probably produced by the passage of active magmatic solutions through the rock, both aiding in recrystallization of the original constituents, and depositing the materials held in solution when conditions of temperature or agents of precipitation were favorable.

It is possible that in some cases the monazite in pegmatized gneiss is formed by the gathering together of the proper elements disseminated through the original rock during recrystallization. It is probable that pegmatization in which much quartz with but little feldspar has formed represents a phase of recrystallization, in which the quartz may have come, either partly or wholly, from the original rock itself, or may have been added by solutions passing through the formations.

In either case, the materials do not represent the work of active magmatic solutions or magmas such as might give rise to regular pegmatite bodies. In those recrystallized or pegmatized rocks where the feldspathic component of pegmatite is not plentiful, monazite occurs but sparingly. On the other hand, monazite is found more abundantly in pegmatized-rock formations in which feldspar plays a prominent part. The common proximity of this form of pegmatization to granite masses, or its gradation into pegmatite bodies, gives evidence of its formation through magmatic agencies.

The monazite of rock-formation has, then, probably been derived from aqueo-igneous solutions, such as give rise to certain forms of pegmatite, and which have in these cases affected large masses of rock.

IV. Commercial Deposits.

1. *Placers.*—The commercial deposits of monazite occur in the gravel-beds of creeks and streams and in the bottom-lands

adjacent to them, Fig. 3. The thickness of the gravels ranges from 1 ft., including over-burden, to 8 ft. The distribution of the monazite in them is, as with all heavy minerals, richer near the bed-rock and poorer above, grading into the over-burden. In some deposits the whole thickness of the gravel, with the finer alluvium at the surface, is rich enough to be washed directly or sluiced down and washed. The extent and value of these deposits vary with the topography of the country and the nature of the gravels. In some places the bottom-lands, containing rich monazite-bearing gravels, are over 300 ft. wide, and extend a half a mile or more along the streams. In other places the bottom-lands are small, and there is but little more than the stream-gravels present. The best deposits are more commonly associated with light-colored gravels and sands, containing considerable quartz débris and fragments of other light-colored rocks, such as pegmatite, granite, mica, and cyanite-gneiss. On the other hand, the absence of much quartz and pegmatitic or granitic débris from the gravels is generally characteristic of low-grade deposits of monazite. The presence of black sands—magnetite, ilmenite, hornblende, etc.—in the gravels does not necessarily indicate a low-grade deposit, unless quartz and pegmatitic minerals are lacking also. Monazite-deposits, in regions where hornblende rocks are abundant, generally contain a large percentage of black sands, and it is then often difficult to concentrate the monazite to a marketable grade. As an off-set to this, however, especially in regions where granite is associated with the hornblendic rocks, gold is often found in the concentrates in quantity more than sufficient to pay the cost of separation, and in the same localities the concentrates generally carry also a quantity of zircon. This zircon is in the form of small, clear crystals with brilliant luster, which range in size up to 1 mm. square and about 2 mm. long.

2. *Residual Deposits.*—It has been found profitable to sluice down and concentrate the surface-soils on the lands adjoining some of the richer monazite-bearing deposits. The residual soils that have suffered but little displacement on the surface can thus be washed profitably to a depth of 3 or 4 in., and where the drift-soil has collected on the gentle slopes below a steeper hill-side, several feet can be sluiced down in some cases. The partial concentration of monazite in the top layer of soil is

caused by the washing away of the clay and other light decomposition-products of the rock. The supply of monazite in the stream-gravels in favorable areas is often replenished by the wash from the hill-side soils during rains, especially where the hills have any considerable slope and the land is cultivated. Under such conditions the stream-gravels are often worked two or more times in a year.

The saprolite or rotted rock underlying the richer deposits of monazite is at some places sluiced down to depths of from a few inches to a foot or so, along with the overlying gravels. At other places, small amounts are removed and washed separately for the monazite they contain. The formations that have been found especially favorable for such work are highly-pegmatized gneiss or schist. Such deposits have generally soon been lost or have grown poor, probably due to the fact that the miners cut through the richer bed or failed to follow it in the direction of its extension. The occurrence of monazite in saprolite is merely an altered phase of the occurrence in hard-rock formations.

With the exception of the plant mentioned above, all the monazite mined in the Carolinas has been obtained from gravel-deposits which lie in and along the stream- and creek-beds, where the monazite is collected after having been liberated from the rocks by their alteration and erosion. While no accurate record has been kept of the percentage of monazite in these gravel-deposits, yet it is undoubtedly true that the quantity present, reckoning from surface to bed-rock, does not exceed 1 per cent. This amount, however, is sufficient to make mining profitable. In many localities it has been the custom to sluice not only the gravels but all the over-burden, inasmuch as even the top soil carries a small amount of monazite.

There are no large hydraulic plants in operation, but nearly all the monazite is obtained in sluice-boxes fed by hand, Figs. 3 and 4. These boxes are fitted at their upper end with a sieve or shaking-hopper, with mesh of about No. 12. The boxes vary in length from 5 to 20 ft., and in some instances are fitted with riffles holding mercury for catching the gold. An interesting fact noted in connection with the deposition of monazite in the stream-beds is that when the gravels have been washed for monazite, and then left for a few months or a year (especially if there has been considerable rainy weather), there is

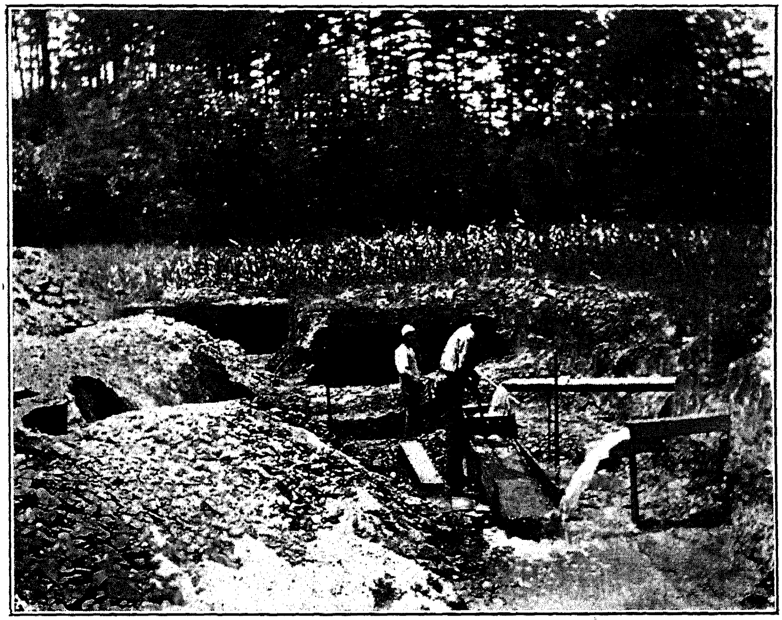


FIG. 3.—MONAZITE PLACER-DEPOSIT, SHOWING LOCATION OF GRAVEL-BEDS IN THE BOTTOM-LANDS AND METHOD OF SLUICING.

FIG. 4.—WASHING MONAZITE GRAVELS.

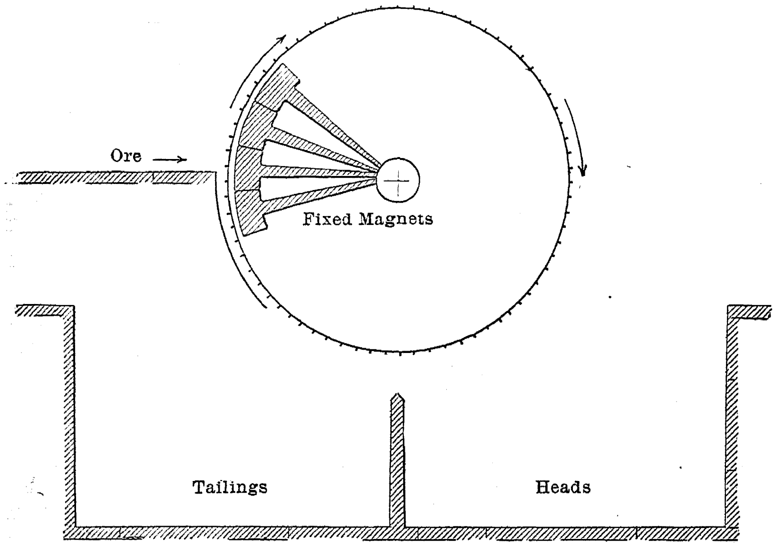


FIG. 5.—SECTION OF THE HEBERLI DRY MAGNETIC SEPARATOR.

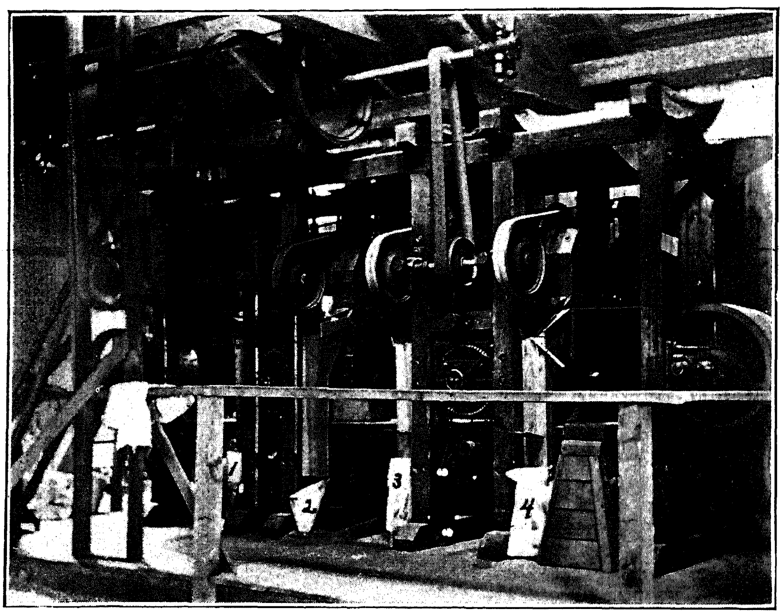


FIG. 7.—VIEW OF A WETHERILL MAGNETIC SEPARATOR.

another supply of monazite deposited, which in many cases can be profitably worked. This monazite has resulted from the washing-in of the mineral from the surface adjoining the streams, where it had been left during the decomposition and erosion of the original rock-matrix. This second deposition of monazite is facilitated by plowing the adjoining fields. In a few places Wilfley tables have been introduced for treating the concentrates from the sluice-boxes. Where these tables are used the soil and gravels are washed into shaking-hoppers and then through sluice-boxes, the over-size thrown out, and the sands fed to the Wilfley tables. At one mine it is necessary to raise the gravels by a mechanical elevator in order to bring them to a sufficient height to feed them to the table. They are fed into a revolving screen and from that to the table. The heads from

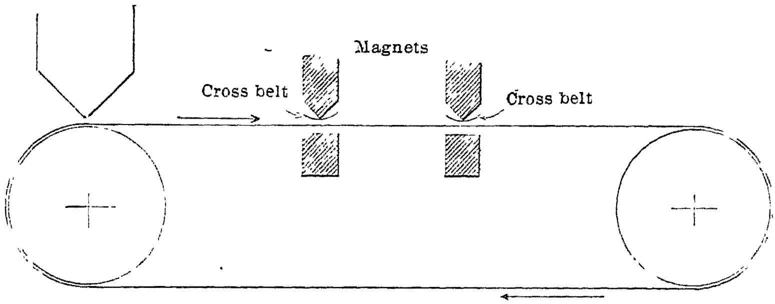


FIG. 6.—SECTION OF THE WETHERILL MAGNETIC SEPARATOR, ROWAND TYPE.

this first washing do not contain a very large content of the monazite, and the middlings are, therefore, re-fed to the table with other feed-ore. In some cases the feed-ore is all run over the machine and a rough concentrate first obtained and then this re-fed. The product from these machines contains from 30 to 70 and occasionally 80 per cent. of monazite. Where there is a large amount of the heavy black sands occurring in the gravel with the monazite, it is almost impossible to get the concentrate much higher than 30 per cent. of monazite. Where, however, these sands occur more sparingly, it is possible by this method to obtain a concentrate containing 70 per cent. of monazite.

All the concentrates from the sluice-boxes and Wilfley tables have to be dried before they can be treated on the magnetic separators. Two different methods are used in the monazite-

district for this purpose. In one the sand is spread over an oiled or a rubber cloth in a thin layer and exposed to the heat of the sun. It dries very quickly, due perhaps partly to the heat absorbed by the dark, iron sand. It requires, however, a considerable surface to accommodate any large amount of sand. The other method of drying is by heating over furnaces. A small ditch, from 4 to 8 ft. long and from 1.5 to 2 ft. wide and about 1 ft. deep, is dug, at one end of which there is built a rock or brick chimney. The ditch is usually built up of stones, with an opening at the end opposite the chimney for firing. Over the ditch there is a sheet-iron cover or drying-plate. The monazite is spread on this plate and exposed to the action of the hot fire underneath. The dried sands are occasionally further concentrated by means of the ordinary horse-shoe magnet, which picks out all the magnetite. The miners are paid for the sand on the basis of 100 per cent. of product, and the higher the concentration, the better the price they receive. The sand brought in to the magnetic-concentration plants is worth from 4 to 8 cents per pound, while after a magnetic separation its value is increased to from 12 to 20 cents per pound.

This material represents what is known as crude monazite sand and contains, besides the monazite, magnetite, ilmenite, garnet, zircon, rutile, corundum, cyanite, hornblende, and occasionally chromite. In order to separate the monazite from its associated minerals, it is necessary to run this crude sand through some electrical apparatus. Two types of machines are in operation: (1) the Wetherill electro-magnetic machine and modifications of this; and (2) machines in which the minerals are deflected by electro-magnets while falling. Of these, the first type is the one most generally employed. By means of these various machines a product can be obtained varying from 90 to 98 per cent. of monazite, and represents the sand that is shipped to the manufacturers of incandescent mantles.

V. Magnetic Separation.

The first application of magnetic separation was in the concentration of certain iron-ores, principally magnetite, in order to produce a product richer in iron, and also to eliminate certain minerals that contained elements injurious to the metallic iron. The next application was to other iron-ores, such

as limonite, hematite, and siderite, after they had been given a preliminary roasting to convert them into the magnetic oxide. The next step was in the separation of magnetic iron particles from certain copper-, gold-, and zinc-ores, either before or after roasting. For many years this was the only application made of magnetic separation. It was found, however, upon experimenting with an electro-magnet with a higher intensity, that other minerals were subject to magnetic attraction, and that it was possible to separate minerals into more or less pure products by varying the intensity of the magnetic field. Thus, it has been possible to adapt this method of separation to ores containing iron or manganese which are only weakly magnetic. As is well known, steel bars may be magnetized, and they will retain more or less of this magnetism indefinitely, while bars of softer wrought- or cast-iron may be magnetized by means of electric currents in surrounding coils of insulated copper wire. These iron bars do not become permanent magnets, but form electro-magnets as long as the current flows around them. They can be given a greater and more constant strength than can be given to the permanent steel magnets, and for this reason, in nearly all of the magnetic processes, electro-magnets are used instead of the field magnets.

The magnetism of these electro-magnets can be varied and different intensities obtained, ranging from indefinitely weak to a certain maximum of strength. It is also possible to control the intensity of any magnetic field, so that minerals that are strongly attracted may be separated from minerals that require a magnetic field of much higher intensity. This intensity of the magnetic field depends:

1. On the size of the magnet.
2. On the shape of the magnet.
3. On the distance between the magnet and the body to be attracted.
4. On the number of ampere-turns in the magnet-coil; that is, the product of the amperes or current flowing in the coil times the number of turns around the core.

There are many substances that are attracted by electro-magnets that are not influenced apparently at all by the strongest steel magnet, and for this reason, many substances which formerly were considered non-magnetic have been proved to be

magnetic when subjected to the intense magnetic field obtained in an electro-magnetic separator. All substances are either attracted or repelled by magnets, and the former are called para-magnetic and the latter dia-magnetic. The latter class is the most numerous, but since the introduction of electro-magnets, the former class, which up to that time had been considered extremely small, has been largely increased. The para-magnetic substances are the metals iron, nickel, cobalt, manganese, chromium, cerium, palladium, platinum, osmium, and many of their salts and compounds. The degree of attraction of these varies very widely, and, as an illustration between a strong and a weak magnetic substance, it has been estimated[5] that if the attraction of steel be taken at 100,000, then magnetite would be 65,000; siderite, 120; hematite, 93 to 43; limonite, 72 to 43. By using the electro-magnetic separators, which can be regulated so as to give a very strong field, and at the same time a field which is capable of fine adjustment, it is now possible not only to separate the para-magnetic from the dia-magnetic substances, but also to separate the para-magnetic substances from each other.

There are three general classes of these magnetic separators: (1) those in which the magnetic particles are held to revolving cylindrical rolls or drums, within which are magnets; (2) those in which the magnetic particles are carried by conveying-belts or pans passing over the magnets; and (3) those in which the ore falls in front of a magnet. There are a number of points of difference in the machines, such as permanent or electro-magnets; treating the ore wet or dry; magnets acting continuously or intermittently; and the use of direct or alternating current. It will be found that different machines are suited for different purposes, according to the character of the material to be treated. As stated before, most of the machines were originally designed simply to treat iron-ores, or to separate iron-minerals from other ores, and there are but few of them that are adapted for the separation of monazite, zinc-minerals, etc.

Class 1.—The Ball-Norton separator[6] consists of two revolving drums, within each of which is a series of stationary electro-magnets so wound that opposite poles are adjacent to one

[5] *Ore-Dressing*, by R. H. Richards, vol. ii., p. 796 (1906).

[6] *Trans.*, xix., 187 to 194 (1890–91).

another. The capacity of a machine with two drums, 2 ft. in diameter and 2 ft. face, is from 15 to 20 tons per hour, with material of from 16 to 20 mesh. The ore is fed upon the top of the first drum, and the magnetic particles are held by the drum, while the non-magnetic fall into the hopper below. As the drum revolves the magnetic particles get beyond the magnetic field and are thrown by centrifugal force on to the second drum. This drum, which does not have quite so strong a current as the first, does not attract the weaker magnetic particles, so that these drop off into a second hopper, forming a middlings product, while the stronger magnetic particles are held by the drum and carried a certain distance, when they get beyond the magnetic field and are dropped into a third hopper. On account of the alternate polarity of the adjacent magnets, the particles roll over and thus facilitate the elimination of any gangue particles that may be mixed with the magnetic material.

Another simple drum separator is the Heberli,[7] which is shown in Fig. 5. In this machine there is but one drum, and the electro-magnets extend over about one-quarter of the area of the drum. The ore is fed to the drum just above the center radius and about the middle of the magnets. The drum revolves in the direction opposite to the feed, and the magnetic particles are attracted by the drum and carried up and over the magnets, while the non-magnetic particles drop into the hopper below. As the magnetic particles leave the magnetic field, they are dropped on the opposite side of the drum into another hopper.

Class 2.—It is principally magnetic separators of the second class that have been used in the separation of monazite in the Carolinas. Of these machines, the Wetherill[8] stands out most prominently, and was probably the first to treat weakly-magnetic materials commercially. The principal idea of these machines is to secure a very strongly magnetic field by concentrating the lines of force as far as possible, this being accomplished by placing the two poles of the magnet facing one another with a minimum air-gap between them and by beveling down the pole-pieces to their end.

The type of the Wetherill magnetic separator that is more

---

[7] *Ore-Dressing*, by R. H. Richards, vol. ii., p. 799 (1906).

[8] *Trans.*, xxvi., 357 to 370 (1896).

generally used is known as the Rowand[9] type, Fig. 6, which has a magnetic pole with sharp edge above the traveling feed-belt and a blunt pole directly under it. Both of these poles are capable of being magnetized by an electric current which will produce a condition varying from weak to intensely strong magnetism. The concentration of magnetism at the sharp edge causes all the magnetic grains to jump to the upper pole. A cross-belt directly beneath this pole, which is running at right angles to the feed-belt, and is running rapidly, readily takes off these grains and deposits them in a bin, while the non-magnetic grains go on with the feed-belt. There can be readily arranged above the traveling feed-belt a series of such poles, each stronger than the one before, so that the first will take off the strongest magnetic particles. The feed-belts used vary in width from 12 to 18 in. The material fed to the machine is classified and allowed to pour over a revolving drum, which concentrates it evenly over the feed-belt. The pole-pieces are made of soft iron, and weigh up to 90 lb. each. They are adjustable, so that the length of air-gap between them may be varied. The strength of the current in amperes can be varied, and also the distance of the feed-belt beneath the poles.

The machines that have actually been in use in the monazite-field are shown in the illustration, Fig. 7. The monazite sand, which is fed to the traveling feed-belt, passes along under four powerful electro-magnets. The first removes all the magnetic iron, and generally all of the titanic iron, or ilmenite, and any chromite that might be present. The second magnet removes all the fine grains of garnet, the coarser ones, if present, usually being removed by the first magnet. The third magnet is so adjusted as to remove only the coarser particles of monazite, while the fourth removes all the finer pieces of monazite. The remaining portion of the sand, consisting largely of zircon, quartz, and a little rutile, corundum, cyanite, etc., is dropped off at the end of the large belt into the waste-pile.

Another type of machine is shown in Fig. 8. In this there are a series of magnets, over which are traveling-belts, which pick out different minerals, according to the intensity of the magnetic field. In this machine the magnetic particles are carried over and under the magnet and dropped into a hopper as

[9] *Ore-Dressing*, by R. H. Richards, vol. ii., p. 807 (1906).

they leave the magnetic field, while the tailings are dropped into another hopper and fed to another traveling-belt and over a second magnet of stronger intensity, which picks out the garnet. This is dropped into a special bin and the balance into another hopper and fed to a third magnet, which picks out the monazite. It is possible by these separators to obtain a monazite sand of from 90 to 99 per cent. of monazite, according to the care that is taken in separating it.

The other products, as the iron-minerals magnetite and ilmenite, and garnet, can also be obtained in a very pure state. From a long series of experiments it has been determined that with machines of this type magnetite can be removed when the amperage is 0.2, ilmenite with 1.1, chromite with 1.6, garnet with 1.75, hypersthene and olivine with 2.2, and monazite with

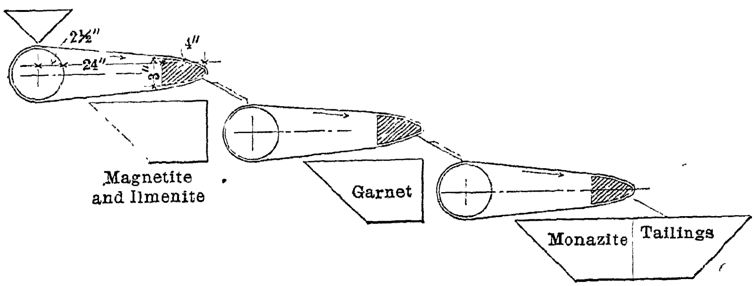


FIG. 8.—SECTION OF THE WETHERILL MAGNETIC SEPARATOR.

3.50 amperes. Zircon is left behind with the gold as non-magnetic. Any platinum that might be present would begin to be lifted by the weakest current, but most of it would not be lifted until the current was 1.5 amperes.

It is possible to separate almost completely pyrite from hornblende by picking out the hornblende with the electro-magnet, the pyrite remaining in the tailings. Such minerals as pyroxene, epidote, titanite, tourmaline, and serpentine are readily picked out by the Wetherill magnetic separator with a current of from 2 to 2.5 amperes. Brookite and cassiterite can occasionally be picked out with an amperage of 3.5.

Class 3.—Perhaps the simplest of all magnetic separators is the one devised by Edison. In this separator the particles of mineral are permitted to fall in a thin sheet in front of the poles of a strong bar electro-magnet, which causes a deflec-

tion of the magnetic particles from a direct downward path, while the non-magnetic particles are not influenced by this attraction and fall vertically. It is possible to make two and sometimes three products in this way.

VI. Uses of Monazite.

The commercial value of monazite depends upon the incandescent properties of the rare-earth oxides which it contains, such as cerium, lanthanum, didymium, and thorium oxides, which are used in the manufacture of the Welsbach and other incandescent gas-light mantles. It is the thoria that is used in largest amount and which gives the actual value to the monazite. In the reduction of the monazite sand there are a number of rare-earth salts that are obtained in considerable quantity, which has made it possible to carry on an extensive series of experiments with these rare-earth oxides. It requires from four to six months to recover from the monazite sand its content of thoria and render it sufficiently pure to be used in the mantles.

The Welsbach mantle consists of a cylindrical hood composed of a net-work of the rare earths, the top of which is drawn together and held by a loop of asbestos or platinum wire. When in use, this mantle is suspended over the flame of a burner constructed on the principle of the Bunsen burner, in which the heating- instead of the illuminating-power of the hydrocarbon of the gas is used by burning it with an excess of air. In this manner the mantle becomes incandescent and glows with a brilliant and uniform light.

A short description of the method of manufacture of these mantles may be of interest. The first part of the process is the selection of the thread or fiber from which the mantle-fabric is knitted. The fiber mostly used is cotton, either the upland, river bottom, Peeler, Allen seed, Sea Island, or Egyptian variety, the market-price varying from about 10 cents for the upland to 30 cents per pound for the Egyptian. The cheaper cottons are used in the lower-grade mantles, the highest-grade mantle requiring the best quality of cotton. The thread is purified, so as to remove every possible trace of mineral matter. If the thread used shows a mineral impurity exceeding 0.15 per cent., it will introduce factors that will affect the physical and

lighting life of the mantle. Cylinders of net-work of various diameters are knitted out of the thread and then washed in ammonia and distilled water and wrung out in mechanical clothes-wringers. After drying, they are cut into pieces sufficiently long to make two mantles.

These knitted fabrics are then placed in a suitable vessel and covered with the "lighting-fluid," in which they remain until thoroughly saturated. The excess of fluid is drawn off and the fabric put through an equalizing-machine, piece by piece. The "lighting-fluid" is composed of a solution of approximately 99 per cent. of thorium nitrate and 1 per cent. of cerium nitrate in distilled water, in the ratio of 3 parts of water to 1 part of mixed nitrates. The fabric is dried and then cut to the proper length required for a hood. It is then shaped over a wooden form and the upper end drawn together by means of an asbestos cord (occasionally of platinum). After the mantle has been modeled the cotton fiber is eliminated by heating the hood over a hot Bunsen-burner flame, leaving the mantle composed of a residue of thoria and ceria. The peculiarity of these oxides is that they have sufficient cohesion to hold together during the remainder of the process of manufacture, after every bit of the supporting cotton thread has been burned away. The hood is then subjected to a series of tempering- and testing-heats, during which it is carefully shaped to its permanent form. In order to protect the mantle during its inspection, packing, transportation, and installation, it is dipped in collodion. Just before using the mantle this collodion covering has to be burned off. It is estimated that the American market consumes 40,000,000 of these mantles per year.

Another element obtained from the monazite is didymium, the oxide of which is dark brown. Use is made of this for branding the mantles with an indelible brand. A nitrate solution is made and an ordinary rubber stamp used for branding.

Of the associated minerals, zircon has a commercial value of from 20 to 25 cents per pound for its zirconia content, which is used in the manufacture of the glower of the Nernst lamp. The fundamental principle of this Nernst lamp is that certain of the rare earths or refractory oxides will conduct an electric current and glow after they have been heated to redness. This discovery, which was made by Dr. Nernst, in 1897, has resulted

in the development and perfecting of the glower which is now embodied in the Nernst lamp. This glower is composed of a mixture of the rare-earth oxides, and is made in the form of a small rod, or pencil, of chalk-like material, having wire terminals at either end. When cold, the glower is an insulator, but it becomes heated to redness when a current is passed through these wires, and its resistance gradually decreases until it has reached a red heat, when with 220 volts across the terminals it starts to conduct the current and give light.

In bringing a glower up to its starting-point, corresponding to a temperature of 1,200° F., use is made of a small electrical heater composed of two or more small tubes, of porcelain, about 1.5 in. long and 0.25 in. in diameter, which are over-wound with fine platinum wire, this in turn being held in place and protected from the intense heat later generated by the glower by an outer coating of porcelain-paste. After the glower becomes heated there is, of course, no further use for the heater, and it is cut out by a small electro-magnet cut-out, which consists of a magnetic coil connected in series with the glower, an armature, and the necessary contacts in the heater-circuit. Thus, when the glower has become heated sufficiently, the current begins to pass through it, and when this becomes sufficiently strong the armature is attracted and the contacts are separated, thus disconnecting the heater from the line. The surface of the glower before being used presents a smooth, white, porcelain or chalky appearance, but after being in use about 500 hr. it is rough or crystalline in appearance.

The yttria used in the manufacture of the Nernst glower is obtained principally from the mineral gadolinite, which has not thus far been found in North Carolina. There are, however, a number of minerals containing yttria, such as samarskite, euxenite, and fergusonite, which have been found in the State.

The magnetite and ilmenite may find a use in the manufacture of magnetite electrodes that are manufactured by the General Electric Co.

The garnet grains are sharp, and can be used for abrasive purposes, especially in the manufacture of garnet-paper.

A Reliable Steel Rail and How to Make It.

BY JAMES E. YORK, NEW YORK, N. Y.

(New Haven Meeting, February, 1909.)

General Discussion.

At a meeting of the American Society for Testing Materials at Atlantic City, June, 1908, Dr. C. B. Dudley, in his presidential address,[1] showed the vital necessity of not only making a steel rail as good as in the past, but of making it considerably better in order to meet the present and prospective requirements of railroads.

Doctor Dudley requested the co-operation of all who from personal experience could give information that would result in bringing about the production of a steel rail that could be relied upon to meet the present requirements.

I have had considerable experience in the practical manufacture of steel of almost every grade, during the past 40 years, and I shall suggest changes that will bring about the results desired. Doctor Dudley has had a long experience as chemical and metallurgical expert for the Pennsylvania railroad, but he frankly admits that up to the present time both he and other metallurgical authorities lack the practical knowledge of steel-making that is necessary to state correctly the reasons for the unreliability of the steel rails now produced. The necessity is self-evident for more positive practical knowledge on the part of these experts of what is possible and what is impossible regarding the practical manufacturing of steel rails from the ingot to the finished product.

Chemical knowledge in steel-making is far in advance of what it was years ago, and might be said to cover every requirement (in that department) for producing reliable steel. The study of the metallography of steel has been of great value, also the physical testing of steel, but this knowledge will only de-

[1] *Proceedings of the American Society for Testing Materials*, vol. viii., pp. 19 to 39 (1908).

velop its greatest value when associated with the absolute knowledge of the conditions affecting the heating and rolling of the steel tested. These theoretical branches are under the supervision of some of the brightest minds in the country, and their experience, tests, and results are at the service of steel-makers. The fact, nevertheless, remains that rail-steel made at the present time, under the most favorable conditions and with the help of chemistry, metallography, physical tests, etc., does not equal in quality the steel made in the past, when chemistry and metallography were comparatively unknown in connection with steel-making. That proper mechanical treatment is of more importance than chemical composition in producing reliable steel rails is shown by referring to the many cases cited by Robert Job[2] and others of rails that have lasted in service from 30 to 40 years having a chemical composition which would have caused their rejection, as rails, if judged by present chemical requirements.

The principal reasons for the poor quality of steel rails made by present methods are:

1. *Overheating.*—All the leading metallurgical authorities, both practical and theoretical, know that overheating steel of any grade is exceedingly detrimental to its physical quality. I herewith give a few extracts from one of the recent books on the metallurgy of steel.[3]

A (p. 233). "In heating steel for rolling, the lower the temperature the better will be the quality of the product."

B (p. 370). "If steel be heated to a high temperature, say 1100° C. (2010° F.), and then cooled (either slowly or rapidly) without being subjected to strain, it will be 'coarse-grained.'"

C (p. 370). "Even the best quality of steel, if rendered coarse-grained by 'overheating,' will suffer in its valuable properties, and may become quite unfit for use."

Mr. Stoughton also explains how overheated steel can be restored (1) by re-heating to a much lower temperature than the original heating; that is to say, annealing after rolling. This method has never been adopted in practice in rail-metal because it greatly lowers both tensile strength and elasticity; (2) by mechanical treatment of rolling.

---

[2] *Proceedings of the New York Railroad Club*, vol. xvii., p. 514 (1907).

[3] *Metallurgy of Iron and Steel*, by Bradley Stoughton (New York, 1908).

Rolling does not achieve the results desired, as I will show by results obtained in practice. Take, for example, the rolling of a 6- by 1.5-in. iron flat, for the reason that the best results from rolling-action are obtained in this class of section. The thickness of the flat selected is about equal to the thickness of the head of a 100-lb. rail. In the manufacture of the above 6-in. flat, it is necessary to form the center of the pile of re-worked iron (see Fig. 1) that has inherent to itself the necessary physical qualities, for the reason that the penetrating-action in rolling does not extend to the center of the pile itself, and therefore cannot be relied upon to develop uniform quality. If it is impossible to get the desired structure in a

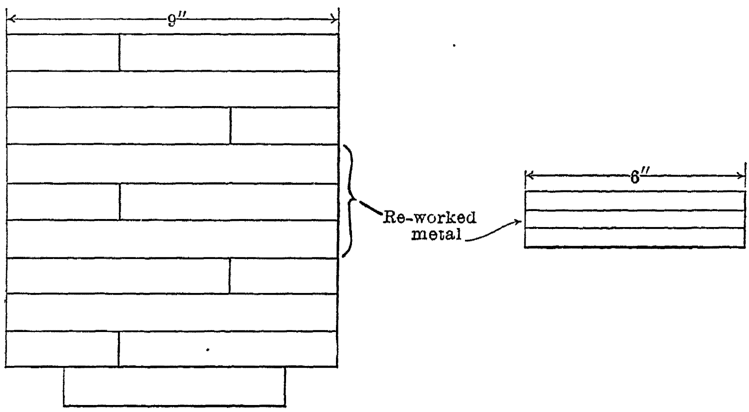


FIG. 1.—PILE WITH RE-WORKED IRON IN CENTER.

mass of metal only 1.5 in. thick, how can it be possible to close and weld pipes situated in the axis of an ingot 20 in. square? This same lack of penetration in rolling applies to all rolled sections in proportion to the mass and the temperature of the bar during the rolling. If you roll the 6-in. bar down to 1 in. the structure improves rapidly; if rolled to 0.75 in. there is no necessity for using re-worked material in the center of the pile, and if rolled to 0.5 in. the physical quality and strength improve more, and so on down to the thinnest sheets. The same results apply to rolling rounds and squares down to wire. The improvement in quality increases as the material decreases in cross-section, with its associated lower temperature.

The rolling of either iron or steel plates shows that thickness and temperature are the controlling factors of the physical qualities obtained by the action of rolling.

In rolling steel plates from an ingot, if the plate is rolled to a thickness of 1 in., it does not have the same tensile strength or fineness of grain that it would have if rolled to a thickness of 0.5 in. Also, when the length of a plate exceeds the width, the extent of this difference will be the measure of physical quality. The width always will be weaker than the length, for the reason that the greater length is produced by rolling when the mass is thinner and the temperature lower, and consequently has received the rolling-action when the conditions were favorable for producing the best physical structure.

It may be stated as an axiom that in proportion to the decrease in rolling-facility, owing to low temperature and reduced mass (factors which generally go together in rolling), the opportunity for improving the physical qualities increases (within well-defined limits).

It is well known that the rolling of armor-plates has been discontinued, for the reason that the rolling-action does not give the same density of texture in the interior as it does in the exterior of the plate.

The examples given prove that in ordinary rolling there is a lack of penetrating-action to produce a bloom of uniform physical structure, and without this uniformity no reliable steel rail, having all the best qualities inherent to the metal itself, can be produced, and consequently all drop-tests are misleading and unreliable.

2. *Solidity of Ingot.*—During my active management of steel-works I bought, on several occasions, a considerable quantity of discarded rail-steel from the tops of ingots. This material had the usual pipes and blow-holes, and I found it impossible to weld these pipes by ordinary rolling, regardless of high temperature given to the metal for this purpose. I then made some experiments on this material, and succeeded in welding it solid by other means, and then rolled it into merchantable steel that was in every way satisfactory.

James E. Howard, Engineer of Tests of the Watertown Arsenal, published an illustrated article under the title, The

Strength and Endurance of Steel Rails,[3] in which is shown a rail-bloom 8 in. square (Fig. 2), rolled down from a 20-in. ingot, that has a lack of uniform structure. Figs. 3, 4, and 5 of the same paper show still more this lack of solidity. In Fig. 5 the finished rail shows seams in the web and a lack of uniform structure in the head and flange. This example illustrates the impossibility of wholly solidifying the ingot by longitudinal rolling.

At the meeting in Atlantic City, above mentioned, an engineer of one of the leading railroads in the United States made the statement that his company had bought 10,000 tons of 100-lb. steel rails, and during the first year in the track 22 per cent. of this tonnage was removed, owing to fractures from split heads, caused, in my opinion, by pipes. His company then placed another order for similar-sized rails, with the proviso that 30 per cent. of the upper part of the ingot should be discarded. The result was that on the second order only 1.7 per cent. was lost in one year from fractures, in contrast to 22 per cent. in the former case. I mention this fact to show the necessity of solidifying ingots before rolling in the blooming-mill, since it cannot be done effectually afterwards by ordinary rolling, and discarding by percentages is very unreliable.

The absolute necessity for solidifying the ingot, before the process of rolling in the blooming-mill is commenced, is apparent from the fact that at no other stage of the process is the temperature high enough or the ingot sufficiently plastic to permit of welding the pipes and blow-holes, and it is impossible, as has been shown by the examples given, for the ordinary rolling-mill to do this work at any stage of temperature, due to the lack of penetrating-action in rolling. Even under the best work this rolling-action can only flatten out the lips of the pipe, and this is the condition in which all rails are finished, unless the entire pipe is cut off with the discard.

A leading steel manufacturer was asked how a solid steel ingot could be procured and how much discard would have to be allowed. He ironically suggested the cutting of the ingot in the middle and throwing both ends away.

3. *Rolling.*—Another cause for unreliable steel rails comes

[3] *Railroad Gazette*, vol. xliv., No. 13, p. 439 (Mar. 27, 1908).

from the fact that as now manufactured a rail is weaker transversely than longitudinally, and this condition is the reverse of the requirements for a good rail. Even when the ingot is solid and the steel not overheated, it would be impossible to produce a steel rail of the same strength transversely as longitudinally, for the following reasons: When the bloom enters the shaping-roll to produce a steel rail, the entire effective compression is all in the direction of the length of the rail, and as it is by this compression that the physical qualities of the rail are developed, the differences in strength can thus be satisfactorily explained. This transverse weakness applies not only

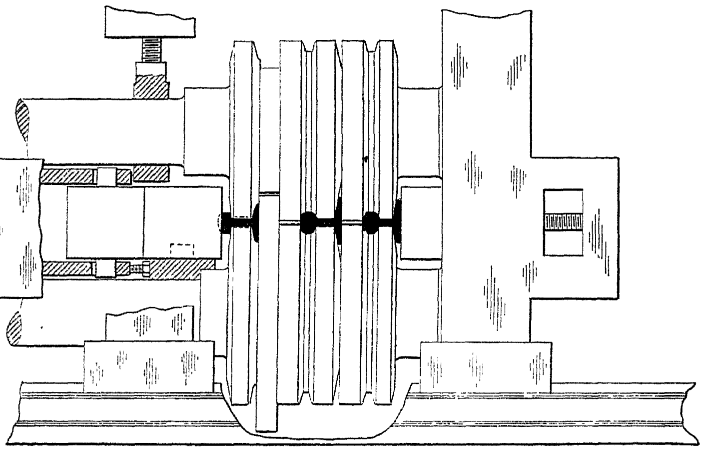


FIG. 2.—YORK UNIVERSAL MILL FOR ROLLING RAILS.

to rails, but to all other flange-sections that are rolled in ordinary mills, and can only be overcome by rolling rails on a universal mill for at least the last three or four passes. A section of the mill showing the passes is given in Fig. 2.

In 1893, I made the first steel beams rolled universally in the United States, and sent these beams to be tested for physical properties to Charles A. Strobel, C.E., Chicago, Ill., the well-known expert in this class of work. He reported that they were the first steel beams he had ever tested that had the same strength and other physical qualities transversely as longitudinally. The importance of this improvement as applied to rail-manufacture will be understood when it is known that the difference in strength varies from 10 to 20 per cent., and is even greater in some cases. The wearing qualities of

the head of the rail would also be vastly improved by finishing the rail at the proper heat, and also compressing the metal in direct lines with the wearing-surface of the head, which is the reverse of present practice. I have given some of the advantages that can be produced in rolling a rail in a universal mill in a paper read last May before the Iron and Steel Institute in London.[4]

It is now suggested by metallurgists, engineers, chemists, and others that the substitution of basic open-hearth steel for Bessemer steel will prove a panacea for the present disastrous results arising from broken rails. The process of making steel ingots, however, can only affect the chemical composition relatively; pipes, blow-holes, and segregations will still have to be contended with. Lower phosphorus in the steel, made possible by the basic process, will undoubtedly provide an ingot from which a better rail can be made than that usually produced by the present Bessemer practice, as it is axiomatic that the smaller the amount of deleterious metalloids in the steel the better the finished product will be, provided that the heating and rolling have been properly done.

Practical Demonstration.

Having briefly stated what I consider wrong in present methods of steel-rail manufacture, I now propose to show how a rail of reliable physical quality can be made without adding anything to the cost. The product, in fact, will be cheapened by saving most of the present wasteful discard proposed in the specifications, and the production of No. 2 rails will be greatly reduced.

Solid ingots are absolutely necessary if good, reliable steel rails are to be produced, and to make a rail-ingot solid is impossible by present practice. The necessity for this condition (solidity) in ingots is incontrovertible. Every leading authority on rail-manufacture since the advent of the steel rail emphasizes the importance of solid ingots. The late William R. Jones, manager of the Carnegie Steel Works, under whose management more steel rails have been made than under that

---

[4] The Physical Qualities of Steel in Relation to Its Mechanical Treatment, *Journal of the Iron and Steel Institute*, vol. lxxvi., pp. 167 to 178 (No. I., 1908).

of any other single individual, said, on this subject, 30 years ago:[5]

"The first thing, in my opinion, toward making a good serviceable steel rail is to make a sound ingot, free from porosity, sponginess or honeycombs, and as hard as is compatible with safety."

Edward Williams, a leading railroad engineer, who is an authority on steel rails, says:[6]

"Let the ingot be as sound as possible, all care taken against overheating (which is a very common source of mischief), . . ."

Dr. C. B. Dudley says, in his recent address in Atlantic City:[7]

"If the ingot is unsound, good rails cannot be made."

The above statements are fully corroborated by the experience of all steel-rail experts.

Solidifying the Ingot by Transverse Compression Before Rolling in the Blooming-Mill.—The best form of ingot, in my opinion, is one in which the corners have a large radius. This form has been recently suggested by P. H. Dudley, Metallurgical Engineer of the New York Central railroad. The reason he gave was, that with more-rounded corners the ingot could be rolled with more-uniform heat; also, that slag would not so readily accumulate there; but the chief advantages, in my opinion, are: 1. It removes, to a large degree, the liability of overheating or burning the corners by the impinging gases in the soaking-pit, and this reduces the liability of cracking during the first passes in the blooming-mill. At this stage of rolling the ingot is always weakest at the corners. 2. It does not present to the rolls as much flat surface for frictional contact, and, in consequence, the reduction of the metal is less severe and the corners are more gradually reduced, and by that time, from rolling-action, the material has become much stronger and tougher and resists the tendency to crack; consequently, there are fewer possibilities of No. 2 rails resulting from physical defects.

All leading experts who have studied the subject are united in the opinion that steel ingots should have continuity of structure, and, as a means to this end, it is suggested that the ingot

[5] *Trans.*, ix., 248 (1880–81). [6] *Trans.*, ix., 248 (1880–81).

[7] *Proceedings of the American Society for Testing Materials*, vol. viii., p. 28 (1908).

should be made so that blow-holes be eliminated, as far as possible, and lack of solidity concentrated in the pipe. It is well known that in tool-steel practice the piped ingot is always selected when reliable steel must be had (the piped part of the ingot being discarded, of course), in contrast to a raised ingot, which is always considered unreliable.

Henry D. Hibbard, Plainfield, N. J., advocates the use of transverse compression to give the continuity of structure in the ingot that is so necessary for reliable steel.[8] As I have for a long time past contended that this was the proper way to achieve the results desired, I take the liberty of quoting from his remarks on this point:

"Granting that continuity of structure is essential in the best steel ingots, the ideal procedure to make them is to start with suitable plant and materials, make the steel properly as to physical and chemical conditions before casting, and reduce or obliterate the pipe and the central segregation by compressing the ingot laterally while the interior is still fluid. It is assumed, of course, that all details favoring these broad divisions of the operation will be adopted. The possibilities of lateral compression have not as yet been fully realized."

As I have attempted to show, the longitudinal or ordinary rolling compresses the exterior of the ingot, and thereby stretches the interior where the pipe is situated, and it is impossible to close or weld the pipe by that method, since the rolling simply stretches the pipe longer and can only result in flattening the lips of the pipe in the later stages of the process. This defective structure remains in the rail and seriously impairs its physical qualities. The process of discarding an equal part of all ingots is wasteful and unreliable, since the length of the pipe varies. In order to remove the necessity for discarding, I suggest the transverse solidification of ingots as the cheapest, simplest, and most efficient method now offered for this purpose.

The practical demonstrations of solidifying ingots by transverse rolling have been given on a mill built for this purpose, a section of which is shown in Fig. 3, and in every case the operation was successful. It not only closes the pipe, but solidifies the entire ingot, giving a uniform structure to the mass. The process is simple, and would not interfere to any extent with the present output of modern rail-mills, but would tend to increase it, since the rolls are short and permit heavy reduc-

[8] *Bi-Monthly Bulletin*, No. 21, May, 1908, p. 422.

tions without springing; and the cost per ton is low (less than one-third of any other known method of ingot-compression).

The operation is conducted as follows: 1. The ingot (or ingots, for quite a number can be treated at one time), is laid horizontally on a table that moves laterally under the rolls at the same surface-speed. The ingot is placed slightly oblique to the face of the rolls, which, by gradually presenting the side of the ingot to the action of the rolls, prevents any shock to the mill.

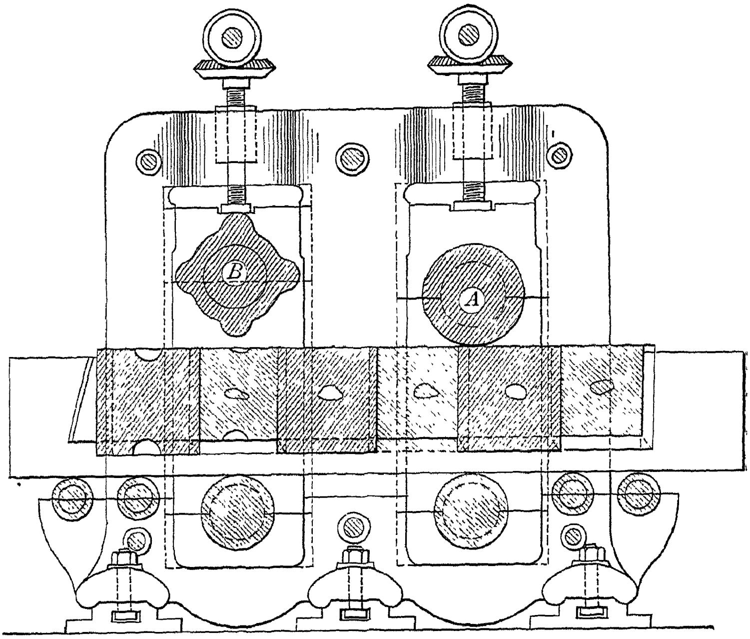


FIG. 3.—YORK TRANSVERSE MILL, FOR SOLIDIFYING INGOTS BY TRANSVERSE ROLLING.

2. There is no material lengthening of the ingot by this method, in direct contrast to the action of ordinary mills, and all the compression is applied to solidify the metal. The first compression raises the segregated matter, if still fluid, which is accomplished by a projection on roll A, that compresses the ingot across the section, in the place where the fluid matter is situated. After this has been done, the compression is applied over the entire face of the ingot in order to close and weld any surface blow-holes that are present, and by so doing prevent, in a large degree, the producing of No. 2 rails. Surface blow-

holes I regard as the chief cause of these inferior rails. The final operation is directed to welding and solidifying the pipe (or pipes) and deep-seated blow-holes. This is accomplished by applying the compression in a direct line with the axis of the ingot by a rib-roll, *B*, that is synchronized with the table, so the pressure is applied at the same place until the ingot is solid. This operation forces the metal from the side into any existing pipes in the interior. The ingots during the operation are so held that there is no material yielding of the metal in any direction except on the side that the roll is compressing, and on the under side, that is pressing upwards in a direct line with the rib-roll.

The entire operation can be accomplished with a very small amount of power compared with other methods.

I have submitted this process to well-known experts in rail-manufacture, and they indorse it for its practicability, quickness of operation, and very low cost of solidifying the ingot.

It is estimated that by this process at least 80 per cent. of the metal it is now considered necessary to discard so as to produce a reliable rail, can be saved for rail-purposes, which, estimated on the entire tonnage capacity of rail-ingots in the United States, would amount to 800,000 tons per year that would be raised from a scrap value of $12 per ton to the finished-rail value of $28—a saving of $16 per ton on 800,000 tons of metal. There would be also a further saving by the elimination, largely, of No. 2 rails. With a solid ingot I see no necessity for rolling No. 2 rails, as a general rule, if a universal mill be used to finish the rail.

The cost of installing machinery to solidify ingots would be comparatively small, and the space occupied also would be small, on account of the character of the work. The machines, consisting of heavy castings similar to ordinary rolling-mills, can be obtained at about the same price per ton. I estimate that a mill capable of treating 60 tons of ingots per hour would cost about $12,000, and weigh about 120 tons. This estimate is for the bare machine, exclusive of power, foundations, buildings, or the other necessary appurtenances for operating; but when it is considered that, by utilizing the discard, the entire cost of installation could be saved every month, it should commend itself to all rail-manufacturers as a very profitable

investment, irrespective of the improved quality of rail produced. The cost of treating ingots should not exceed $0.75 per ton, including royalty to the inventor.

The cost of installing a universal mill for finishing the rail for the last three or four passes would not exceed the cost of an ordinary mill of similar weight. A sketch of a mill of this type is given in Fig. 2. The advantages of this practice are manifold; not only would the physical qualities of the rail be greatly improved, but the mechanical construction of the mill

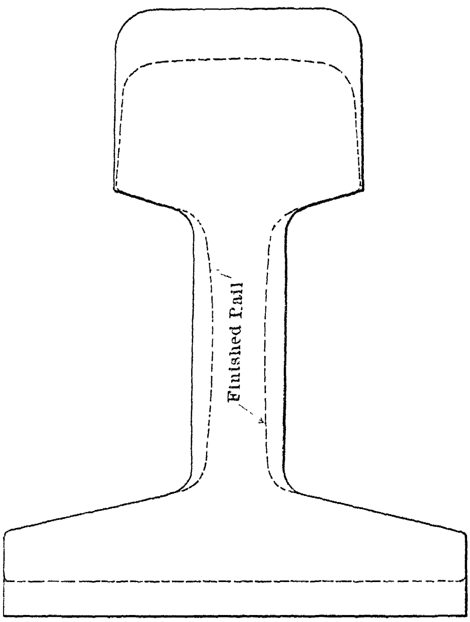


FIG. 4.—BLANK FOR UNIVERSAL MILL.

permits the rolling of the rail at the desired temperature to produce the mechanical hardness so essential to toughness and resistance to wear. Experience has shown that the delivery-speed of the rolls can be greatly increased without leaving in the finished product the internal stresses found in the finished rail made by present methods, which are caused by different speed-deliveries of the different surfaces forming the rail.

The present form of blooming-mills and shaping-rolls can be used. Also, the blank can be properly proportioned to the finished rail desired, as shown in section in Fig. 4.

A practical demonstration, using a small York universal mill, has proved conclusively that the physical properties of the rail are greatly improved.

Registering the Physical Qualities of Steel During Rolling.

During the past three years I have written various papers concerning the improvement of the physical properties of steel rails. In these papers I have outlined the changes necessary in the present practice of the mechanical treatment of the ingot, including solidification, proper heat before rolling, and proper temperature for finishing the rail.

I have devised a process by which the suggested improvements in quality can be controlled, and the physical results registered in the rolled material itself, without any radical changes in the method of rolling.

The recent introduction of electric motors to furnish the power to roll all kinds of steel-sections, and while rolling to register the amount of energy consumed during the operation, furnishes us with the means of arriving at the physical properties of the steel produced from a metal of a known chemical composition.

In the manipulation of steel, heat and work are closely related. Heat displaces work. Efficient work is the measure of quality in the finished steel, so it follows that the more work incorporated in the steel product at proper temperatures the better will be the physical qualities. Consequently, as the heat increases, the steel softens, and this condition displaces work in a relative degree, since the compression of the metal under high heat requires much less energy as compared with the corresponding compression at the proper heat required for the best physical results. The record, in kilowatts, of the power required to roll a given sized ingot into a rail can be made to control the heat of the ingot, and the temperature at which to finish the rail, since any change of temperature is recorded by the motor—more power in proportion to the reduction being demanded when the heat is low and less when it is high. This process will furnish a simple and reliable pyrometer for controlling the temperature of the steel during the rolling, and its application to the manufacture of steel rails and other sections promises many advantages.

The Coal-Mines and Plant of the Stag Cañon Fuel Co., Dawson, N. M.

BY JO. E. SHERIDAN,* SILVER CITY, N. M.

(New Haven Meeting, February, 1909.)

I. GEOGRAPHY.

THE Dawson coal-mines are owned and operated by the Stag Cañon Fuel Co., of which Dr. James Douglas is President and E. L. Carpenter general manager. The property is situated in Colfax county, N. M., and the openings now in operation are in townships 28 and 29 N. R. 20 E., and township 28 N. R. 21 E., shown in Fig. 1. The mines are part of the southern end of the Raton coal-field, which extends north into Colorado and embraces many coal-camps of the Trinidad section.

II. GEOLOGY.

Geologically, the coal-measures, commonly known as the Laramie series of the Cretaceous system, have a thickness of about 800 ft. in the vicinity of Dawson.

The coal makes an excellent coke, and, according to some authorities, its coking properties are due to the action of intruded sheets and sills of igneous rock, the sheets occasionally thickening into masses resembling laccoliths, at such places making nearer approach to the coal and sometimes producing small areas of natural coke. There are but few dikes throughout the entire southern portion of the field and little or no faulting along the dikes. In some parts of this great coal-field the intrusive sheets are far removed in the green shales below the coal-measures, and are so much altered as to be difficult of identification. There is but little disturbance of the strata throughout the Raton coal-field in as far as it extends into New Mexico.

There are two workable seams in the coal-measures, and two or three smaller coal-seams, ranging from 1 to 2.5 ft. in thick-

---

* Territorial Mine Inspector.

ness. The Dawson mines are located upon the lower of the two workable seams, which is known as the Raton or Blossburg coal-seam.

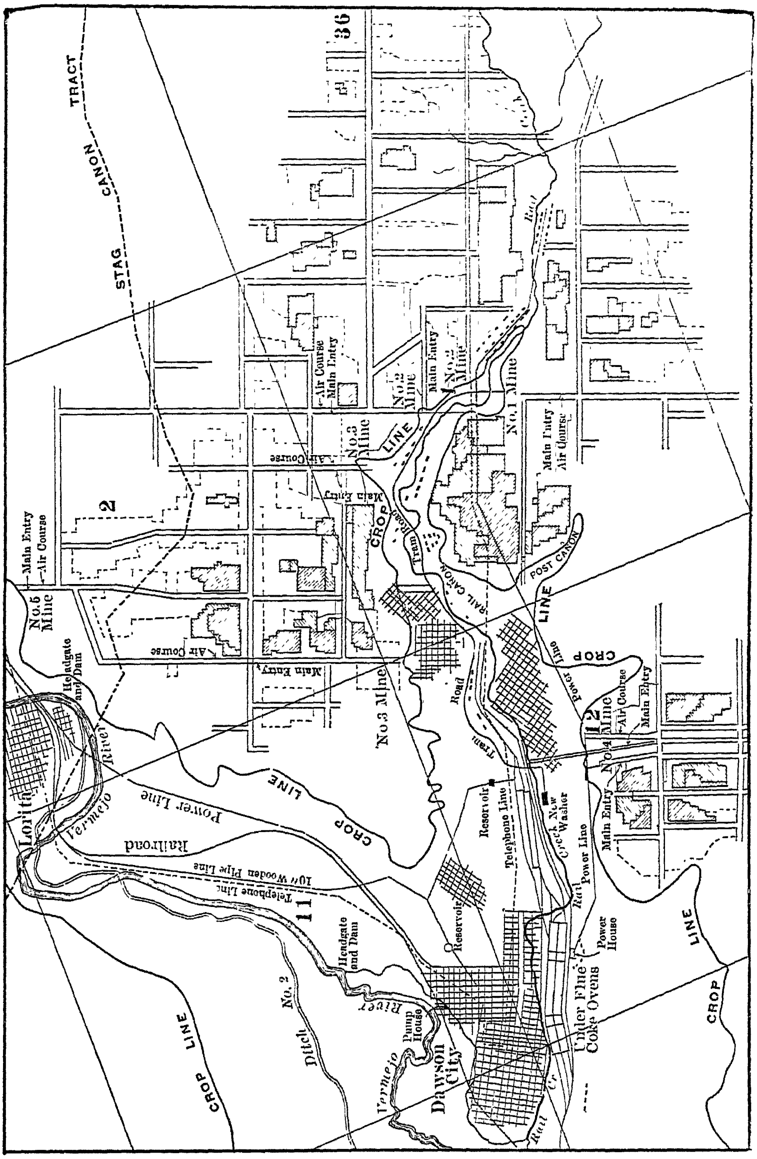


FIG. 1.—WORKING-PLAN OF THE STAG CAÑON FUEL CO.'S MINES NOS. 1, 2, 4, 5, AND 6, COLFAX COUNTY, N. M.

The Stag Cañon Fuel Co. owns about 38,300 acres of land underlain by this great coal-seam. In the Dawson mines the

thickness of the developed seam varies from 6 to 11 ft., with an average of at least 7 ft. Computing the tonnage which may be recovered upon the usual basis of 100 tons per inch of thickness per acre, there are 321,720,000 tons of coal in the property, and deducting 20 per cent., or 64,344,000 tons, for eroded gulches and insurance against unforeseen losses, such as mine-fires, squeezes, and other unfortunate conditions, there remains 257,376,000 tons to be recovered.

III. Mining.

1. *General.*—The topography of the field favors the economical and rapid development of the coal. The eastern projection of the elevated plateau or table-land has been eroded, exposing the green shales below the coal-measures, and leaving a bold escarpment along the entire side, whereon each stratum and coal-seam is distinctly identified. The Vermejo river and a few small cañons or gulches intersect the land in such manner as to expose a crop-line which aggregates a length of about 40 miles. From these exposures the coal-seam may be economically developed by as many openings as are necessary to supply the demand for the product. One of these openings is shown in Fig. 4. At present five openings are in operation, known as mines Nos. 1, 2, 4, 5, and 6. Mines Nos. 3 and 5 were connected by entries more than a mile long, between Rail cañon and the Vermejo river. The consolidated mines are now known as mine No. 5. Mines Nos. 1 and 2, located in Rail cañon, have entries driven into the field for a distance of more than a mile; the coal at the faces shows a thickness of 8 ft. 4 in., and is apparently cleaner than that near the outcrop. All of the mines are opened by drift-entries, which are rendered practicable by the continuous outcrop of the coal and the easy and constant dip of the seam, from N. 10° W. to N. 30° W.

The system of mining is by triple main-entries, double cross-entries, room-and-pillar, and robbing on retreat, when the district becomes exhausted. The width of main- and cross-entries and air-courses is 9 ft.; the height of air-courses, 6 ft. 6 in.; the height of roads, 6 ft.; room-necks, 20 ft.; average width of rooms, 24 ft.; average length of rooms, 350 ft.; distance of room-centers, 50 ft. The coal is hauled by mules from the rooms to the partings within the mine, whence it is brought to

the outside yards by motors, of which there are 10, of the Jeffreys, Westinghouse, and Goodman types. A system of electric signal-lights is used, a red light hanging beside the regular mine-light. As the motor enters each block a red light is turned on automatically to give warning that a car is coming on that block. Mines Nos. 1 and 2 are ventilated by two Vulcan fans, 24 by 8 ft., exhausting, but reversible. These fans are driven by two 50-h.p. alternating-current induction-motors; slip-ring, variable-speed type. There are also auxiliary, direct-current 50-h.p. motors, which can be run independently in case of emergency. Each fan, operating at 60 rev. per min., and a pressure of 1.2 in. water-gauge, produces an intake ventilating-current of about 80,000 cu. ft. per min. Mines Nos. 4 and 5 are ventilated by two Cole 15-ft. diameter straight-vane fans.

The following data, pertaining to the operation of mines Nos. 1, 2, 4, and 5, are of interest: Average number of miners on the pay-roll, 700; average number in the mines each day, 620; number of company men constantly employed underground, including drivers, trappers, timber-men, fire-bosses, motor-men, and pit-bosses, 115; the total air-intake averages 260,558 cu. ft. per min.; 59 mules are used for gathering the coal from rooms to the partings; and allowing 600 cu. ft. of air per min. for each mule, or 35,400 cu. ft. for 59 mules, there remains for the use of the 735 men underground 225,158 cu. ft. of air per min., or 306 cu. ft. per min. for each man employed. The water-gauge varies from 0.8 in. in No. 4 mine, with the shortest pull, to 1.2 in. at No. 2 mine, the longest pull. The air-measurement is given in the aggregate, for brevity, but each mine has its proportionate share for persons underground, which amounts to three times the quantity required under the United States law governing the operation of mines in the Territory.

About Apr. 1, 1909, an air-shaft will be sunk from the surface at a point one mile north from the mouth of mine No. 2. This shaft will be 12 by 12 ft. in the clear, and 250 ft. in depth to the intersection of the main return air-course of mines Nos. 2 and 5. A fan of large capacity will be installed at the top of the shaft, exhausting through the shaft, the present openings to be used as intakes.

From mines Nos. 1 and 2 the coal is conveyed to the tipple

in mine-cars over a tramway 6,600 ft. long, which has a rise of 112 ft. from the tipple to the mines. Six locomotives haul these cars, as follows: Two 28-ton Porters, one 20-ton Vulcan, one 18-ton Lima, and two 6-ton Porters. The tipple is a double Phillips tipple, with two chutes for loading railroad-cars; the tipple-equipment also includes stationary and shaking screens, for sizing coal for various purposes, also a moving slate-picking table.

The coal from mine No. 4, which is located immediately opposite the tipple of mines Nos. 1 and 2, is delivered over a steel Phillips tipple abutting the tipple of mines Nos. 1 and 2. At mines Nos. 5 and 6, the coal is screened as it is unloaded on to railroad-cars, the slack being hauled to the slack-bin, shown on plan of washery (Fig. 3), whence it is elevated to a belt traveling to the washery storage-bins.

A complete telephone-system, having stations at the most convenient points within the mine, affords communication with every important place in the camp, and through the central station with Santa Fé, Albuquerque, Denver, and other cities.

The mines are sprinkled by water-cars to lay the coal-dust, which is removed from the roadways, as far as practicable, and taken out of the mine. Extra fire-bosses have recently been employed at each of the mines to instruct the men in regard to timbering and to see that every precaution is taken to guard against accident from careless work by the miners.

2. *General Rules.*—The following rules and regulations have been adopted by the Stag Cañon Fuel Co. for the government and operation of its mines, and distributed to the employees in convenient pamphlet form under date of Aug. 3, 1908:

1. It shall be the duty of each and every employee of this company to inform himself in reference to his duties under the mining-laws of this Territory and to comply strictly therewith.

2. No person in a state of intoxication shall be allowed on any of the works, or allowed to enter any of the mines, under penalty of prosecution for trespass under the law.

3. No person or persons shall be allowed to enter any mine except he be a regular employee of that mine, or unless he has a permit from the mine-foreman or superintendent.

4. Persons seeking employment shall procure it outside of mine. No boy under twelve (12) years of age shall be permitted to work in any mine.

5. If any person rides upon or in the mine-cars going in or out of the mine or on the tram-road, he does so at his own risk.

6. All persons, except those duly authorized, are forbidden to meddle or tamper in any way with any electric lights, switches, signal-wires, or shooting-wires in or about the mines.

7. No person or persons shall go into abandoned parts of any mine unless permission be granted by the mine-foreman.

8. All persons before entering the mine must deposit a check at check-house, and get the same when they come out of the mine.

Fire-Boss.

9. The fire-boss shall make, before any person is allowed to enter the mine, a careful inspection with a safety-lamp of every working-place in the mine, marking the day of the month on the face of the coal in each working-place where it can be readily seen. If dangerous gases are found in any working-place he will mark on a cap-piece or shovel two large crosses with the day of the month between them, thus: X 27 X, and will place these marks so that it will be impossible for any one to pass them without seeing them.

If a quantity of gas is found, which, in the opinion of the fire-boss, would endanger the operation of the mine, he is authorized to close the entire mine or any part of it he thinks endangered. The fire-boss must always be on the safe side. The fire-boss must not allow gas to be moved where men are working in the return-air from it.

After complete examination of the mine has been made, the fire-boss shall come out of the mine and make a report in Report Book of all dangerous conditions found, which report must be read by the mine-foreman before any men are allowed to enter the mine. The fire-boss shall remain at mouth of mine, or some convenient place, until all the men have entered the mine, instructing each man as to the condition of his working-place.

The fire-boss must make an inspection at least once a week of all old or abandoned parts of the mine and report conditions of same in Report Book.

Mine-Foremen.

10. The mine-foremen shall familiarize themselves with the mining-laws of the Territory, and shall comply with the requirements thereof by discharging every duty imposed upon them by law and by the rules of the corporation.

11. They shall visit each working-place at least once every week and direct the miners and all other employees in their work, and see that their instructions are complied with. They shall direct the miners to securely prop their working-places and see that break-throughs are driven at proper distances. They shall see that the ventilation of the mine is kept in good condition and that all dangerous conditions are removed as soon as possible. They shall have absolute authority over all underground employees, and see that all the rules and regulations are carefully carried out.

Miners and Other Employees.

12. All employees shall use every precaution to prevent accidents in or about the mine; they shall not work in an unsafe place when timber would remedy the danger. If timber is not at hand they must stop work and report the fact to the mine-foreman. The miner shall each day, before beginning work, examine his working-place and take down all dangerous rock, or otherwise make it safe by properly timbering, and shall carefully sprag the coal when undermining.

13. No miner or other employee shall be permitted to burn kerosene, blackstrap, or machine-oil in his lamp.

14. It shall be the duty of every miner to ascertain from the fire-boss the condition of his working-place before entering the mine.

Wireman.

15. It shall be the duty of the wireman to see that all the employees are out of the mine and the power cut off the mine before he enters the mine to connect up shooting-circuits, and to see that all shooting-circuits are disconnected from power lines after shots have been fired; also to see that shooting-lines are kept up in good shape and that miners are furnished wire for extensions, and to see that all wire is removed from pillars and abandoned places.

He shall make daily report in Record Book of the cutting-out and cutting-in of shooting-circuits.

Shooting-Regulations.

The following regulations for drilling and charging shot-holes, mining and cutting the coal will hereafter be in effect at Dawson mines, and must be strictly carried out by all parties:

1. The mining or cutting must extend at least 6 in. beyond back of holes in all cases.
2. All holes must be at least 2½ ft. in length; no shorter holes will be fired.
3. All coal-dust must be extracted from holes before they are charged.
4. No holes must be charged with more than five (5) sticks of powder.
5. Standing-holes, or parts of standing-holes, must not be re-charged.
6. The hole in a tight corner must be at least 1 ft. from rib at back end of hole.
7. In solid faces, holes must not be more than six (6) ft. apart horizontally, and not less than two such holes shall be fired.
8. The object of these rules is to prevent and remove the danger from blown-out or windy shots, and it shall be the duty of the shot-inspectors, in addition to the above rules, to refuse to shoot any holes which, in their judgment, may be dangerous, whether the circumstances are fully covered by the rules or not.

The Taking of Giant Powder into Mines.

9. When giant powder is used in mines not more than fifteen (15) sticks must be taken in the mine for any one working-place for any one shift, and in no place must there be more than twenty (20) sticks at any one time.
10. No giant powder must be taken in the mine in a frozen condition, and any attempt to thaw it out in the mine is strictly prohibited. Miners must have their powder supplied to them at the proper temperature to be exploded. Miners are prohibited from accepting, and powder-men forbidden from giving out, powder in a frozen condition, and shot-inspectors are hereby made responsible for the strict carrying-out of this rule.
11. Giant caps must not be kept in the mine; the shot-inspectors will give them out to the men, one for each shot, as they are needed, and personally supervise the placing of them in the hole with the powder. Under no condition must they be kept with the giant powder.
12. The powder-man will not give giant powder to any person not supplied with a canvas bag in which to carry it.
13. Mine-foremen, shot-inspectors, powder-men, and all others connected with the handling of giant powder going into the mine, must personally see that the above rules are carried out, as far as their supervision in the matter extends.

Powder-House Regulations.

14. No intemperate man or habitual smoker must be employed as powder-man, and, when on duty at the powder-magazine, the powder-man must not have on or about his person, in the magazine, any pipe, tobacco in any form, or matches, nor any tools or materials from which a spark might be emitted or a light created.

15. When powder is being given out to the miners no one but the powder-man must be inside the magazine, and no person must be allowed around the door of the magazine with a light or while smoking.

16. The presence of women, children, or any person under eighteen years of age in or around the magazine is prohibited at all times; also their employment in handling powder, and no powder shall be given out to them.

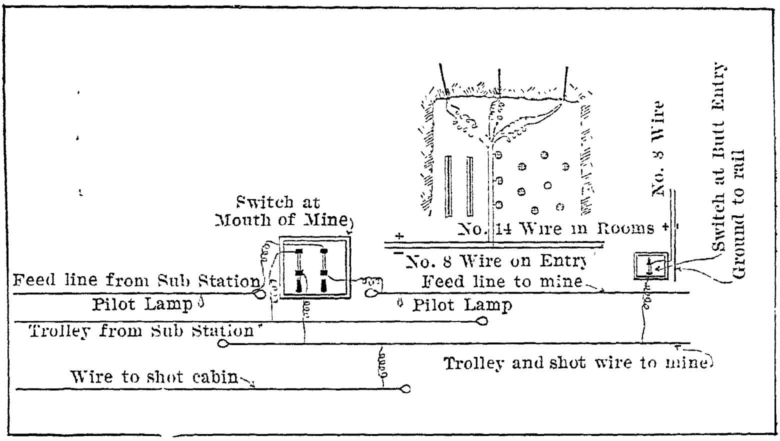


FIG. 2.—DIAGRAM OF SHOT-FIRING SYSTEM IN DAWSON MINES.

3. *Electric Shot-Firing.*—The shooting is done by electricity after all the men are checked out of the mine. As the men enter the mine they are required to deposit a metal check at the shot-firing house outside, near the entrance to the mine. These checks are placed on a check-board and returned to the men as they come from the mine. A record of the working-place of each check-number is kept in the shot-firing house, and in case any check is uncalled for, the shot-firer makes a search for the man until he is found. No shots are fired until it is known positively that no one is in the mine. The method of placing the shots is shown in Fig. 2.

To insure safety against accidental discharge of the shots by electricity, there are two or more locked switch-boxes in each mine, with throw-off switches, one at the mouth of the mine and at one or more stations inside the mine. After inspecting

the inside-connections with the shots to be fired, the shot-firer *en route* from the mine makes connection at each of the switches mentioned. He then goes to the shot-firing cabin to turn on the electric current, but before doing so he turns on an electric signal-light in a red globe, to warn all persons to remain away from the vicinity of the mouth of the mine; so that should an explosion occur within the mine, no one outside could be injured by flying débris. The shot-firing system has proved a success; the safety of the men from disastrous dust-explosions due to blown-out shots is assured; miners make better wages, and the production of coal is proportionately greater per man employed. A record is kept of the number of shots fired, showing less than 2 per cent. of missed shots. The missed shots are left for the next day's shooting, and are either re-primed or a new hole drilled to perform the work intended for the original shot. Very little fire-damp has been encountered thus far in the mines; but a supply of Wolf safety-lamps is kept ready for use.

4. *Safety Precautions.*—A Babcock two-cylinder chemical fire-engine is kept on a side-track, under cover, ready for instant use; also portable chemical fire-extinguishers, and helmets of various types to supply means of respiration in any vitiated atmosphere. Hose-reels, each carrying 500 ft. of best grade of fire-hose, are kept at stations throughout the camp, and a man is employed to inspect daily the hose and fire-fighting appliances.

An organized first-aid corps has had regular practice and competitive drills during the past year, for which the company contributed appropriate prizes and medals for the most efficient team-work.

A large building is being erected for a rescue-station, in which the first-aid corps and others may practice and exercise while wearing the helmets in a chamber filled with vitiated gases. An instructor watches the men, and on showing any signs of exhaustion they will be quickly removed and the gases dispelled from the chamber by suitable outlets. After sufficient experimental work to demonstrate which type of helmet is best adapted to the needs of the mines, a supply will be purchased for use in cases of emergency.

The rescue-station is designed after plans of the one in use

at the mine of the Dominion Coal Co. in Nova Scotia, modified to some extent. In the upper story of this building there will be a technical library on coal-mining, and a "School of Mines" will be conducted by a competent instructor The superintendents, pit-bosses, fire-bosses and others occupying responsible positions in the mines will be required to pass an examination, and if not proficient in the technical and theoretical studies pertaining to their respective positions, as well as in the practical application of these studies, they will be given six months in which to perfect themselves. If, after this time, they are still deficient, they will be reduced in rank or discharged. It is the aim of the company to introduce and maintain such an excellent standard that a certificate to a graduate of the Dawson School of Mines will be recognized as a guarantee of competency.

The powder-magazines at the mines, built of stone, iron, and cement, are absolutely fire-proof. The heat is supplied by electric radiators, which maintain a constant temperature within the magazine; the electric stove or radiator and all wires are at a considerable distance from the stored powder, and out of reach of anything combustible or explosive.

IV. The Coal-Washing Plant.

The coal-washing plant, designed by Dr. L. D. Ricketts, was erected under the immediate supervision of T. H. O'Brien. The plan and elevation of the washery are shown in Fig. 3, and views of the washery-building and storage-tanks in Figs. 4 and 5. The main building, 112 ft. long, 70 ft. wide, and 70 ft. high, and the laboratory- and crusher-building, are absolutely fire-proof, being built throughout of reinforced concrete and structural steel.

Starting at the tipple, the undersize coal from the Nos. 1 and 2 tipple-screens is delivered on a 28-in. cross-belt conveyor, *A*, Fig. 3, running at right angles to the main belt, and driven by a Western Electric motor, 14 h.p., and carried to a 36-in. belt-conveyor, *C*, which is driven by a General Electric motor, 30 h.p. Another 28-in. belt-conveyor, *B*, driven by a Western Electric 14-h.p. motor, delivers the slack from the screens of No. 4 tipple to the same 36-in. belt-conveyor, *C*, and an elevator carries the slack from mine No. 5 slack-bin to join the

undersize from the other mines on the 36-in. belt-conveyor, *C*, which conveys the whole to the two 1,000-ton storage-tanks,

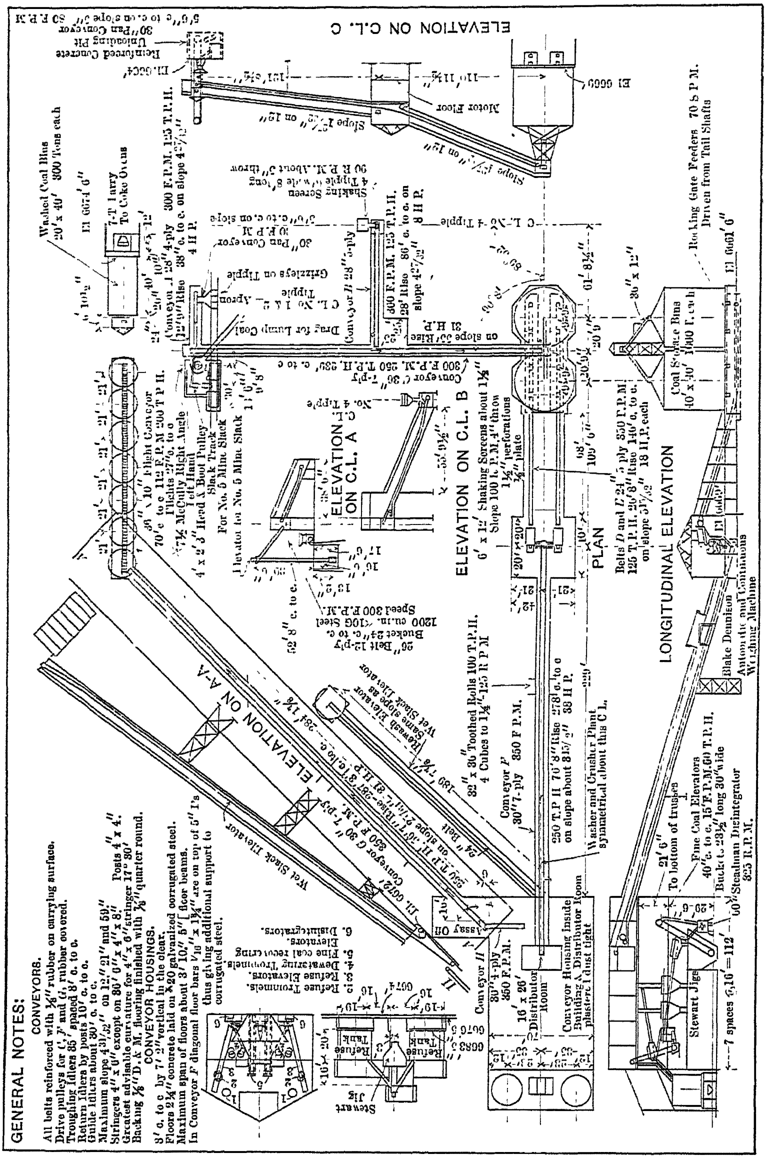


FIG. 3.—PLAN AND VARIOUS ELEVATIONS OF WASHERY.

each 40 ft. in diameter and 40 ft. high. These storage-tanks guarantee a constant supply to the crusher-house and washery, so that they are not dependent upon the work of the tipples.

FIG. 4.—GENERAL VIEW OF WASHERY AND TIPPLES AT MINES OF STAG CAÑON FUEL CO., DAWSON, N. M.

FIG. 5.—GENERAL VIEW OF WASHERY, TIPPLES, AND COKE-OVENS AT MINES OF STAG CAÑON FUEL CO., DAWSON, N. M.

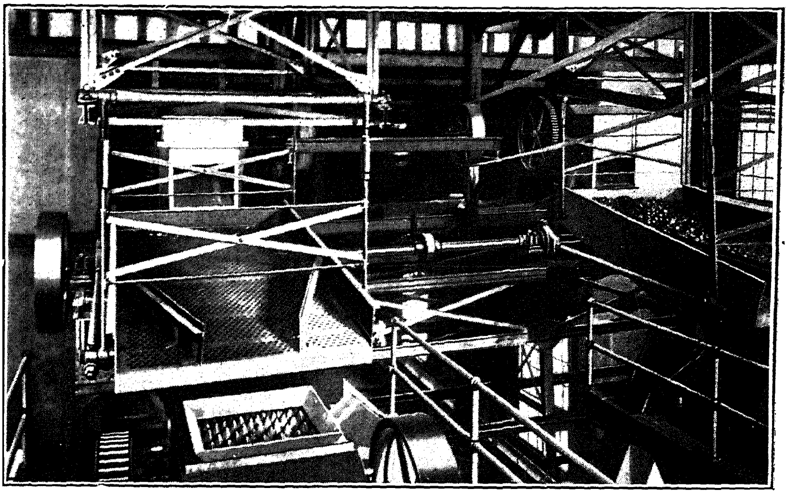


Fig. 6.—Interior of Crusher-House, Showing Screens and Rolls.

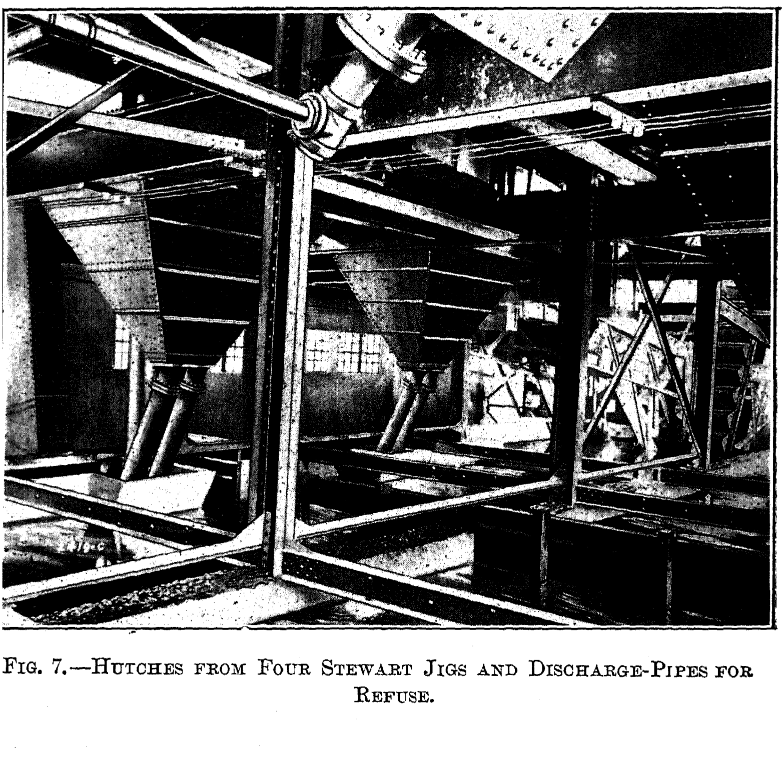


Fig. 7.—Hutches from Four Stewart Jigs and Discharge-Pipes for Refuse.

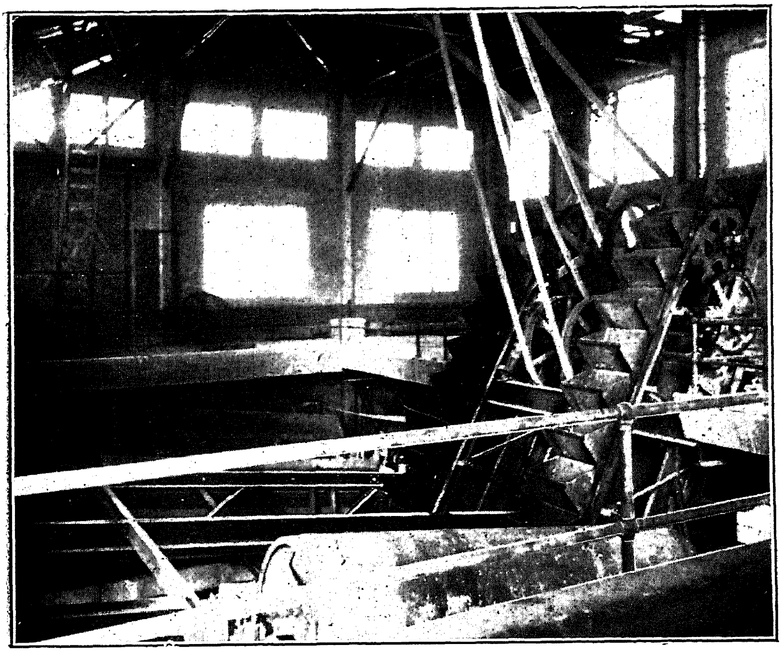


Fig. 8.—De-watering Trommels.

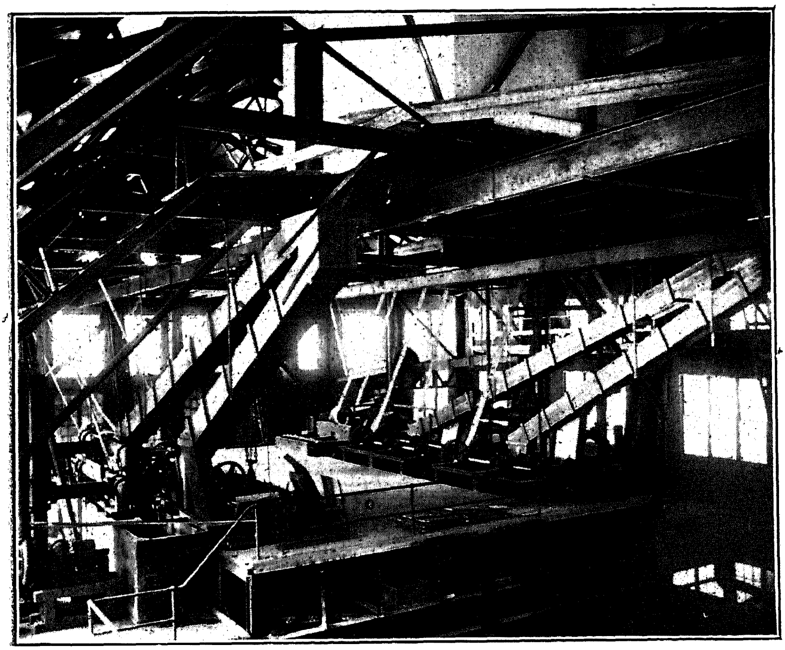


Fig. 9.—Section of Jig-Floor; Refuse-Elevators, from Stewart Jigs, on Right.

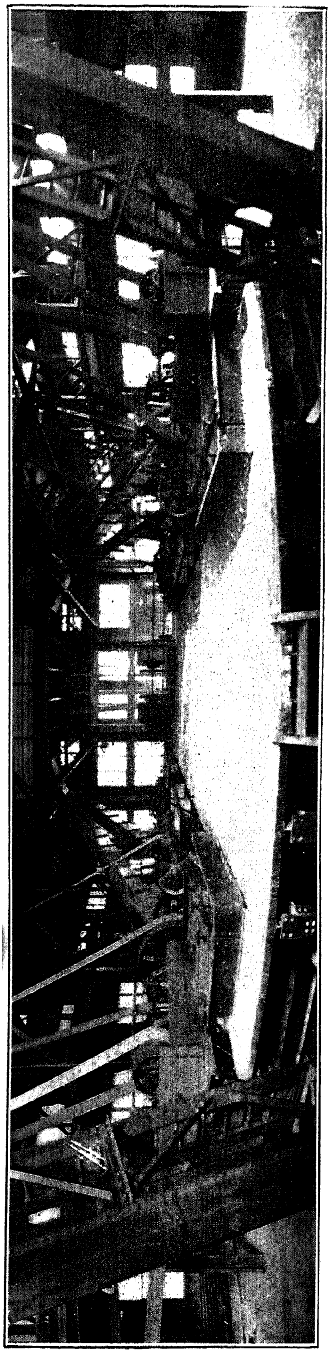


Fig. 10.—Jig-Floor of Washery; Lührig Jigs in Foreground, Stewart Jigs in Background.

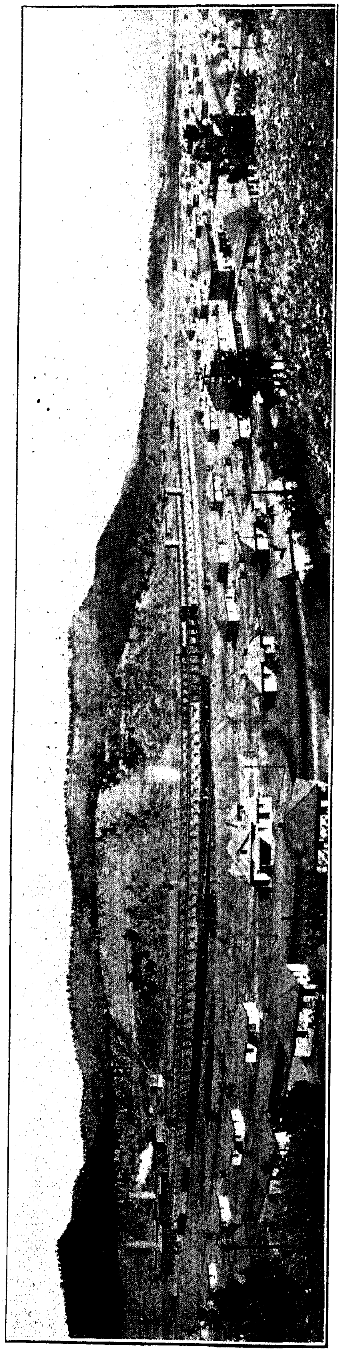


Fig. 11.—Two Parallel Strings of Under-Flue Coke-Ovens, Showing Boiler-Houses between Batteries on Each String of Ovens, and Power-House in Background.

Under the storage-tanks are two 28-in. parallel belts, *D* and *E*, upon which the slack coal is delivered from the storage-tanks, through eight rocker-gate, adjustable automatic feeders, and conveyed by these belts to the crusher-house, where it drops from the belts upon two 6- by 12-ft. shaking-screens, about 1.5 in. slope to the foot, 0.5-in plate, with 1.5-in. round perforations. The oversize is delivered to two 32-in. toothed rolls, 125 rev. per min., 100 tons per hour capacity, Fig. 6, which reduce the material to 1.25-in. size, to correspond to the sizing of the shaking-screen above. The two 28-in. belts and the screens and rolls are driven by an 85-h.p. General Electric motor.

The product from the screens and rolls is deposited upon a 30-in. belt-conveyor, *F*, which carries it to the dust-proof room on the third floor of the washery. As this belt with its load of slack leaves the crusher-house *en route* to dust-proof room, each 25-ft. section is automatically weighed and recorded by a Blake-Dennison automatic and continuous weighing-machine. Thus the data of results are based upon accurate figures. This belt is 278 ft. long, center to center, 76 ft. 8 in. rise, and has a capacity of 250 tons per hour; it is driven by a 50-h.p. Western Electric motor.

In the dust-proof room water is added to the crushed coal by two 5-in. centrifugal pumps driven by two 20-h.p. induction-motors, and the whole is carried in launders to eight jigs of the Stewart type, two double jigs on each side of jig-floor. The jig and water-supply tanks are of steel plate, concrete lined. The pumps which supply water to these jigs are driven by two 50-h.p. Western Electric motors.

From the dust-proof room onward the washery-plant is built in two units, on the east and west sections of the building, and operated independently or together, so that an accident on one side offers no hindrance to the continued operation of the other half of the plant.

The hutches of the jigs, Fig. 7, taper downward, and are connected with two No. 5 Lührig elevators by 8-in. pipes. These elevators discharge the refuse into launders, which deliver it to two refuse-trommels, 4 by 8 ft. All trommels have $\frac{5}{16}$-in. perforations, $\frac{3}{16}$-in. plate, 1.5-in. slope to the foot, and are operated at a speed of 17 rev. per minute.

The oversize from the refuse-trommels passes to re-wash jigs

of the Stewart type; the undersize is re-washed in four Lührig jigs, two on each side; the recovery from these jigs joins the washed coal from the primary Stewart jigs, and is conveyed by launders under the jig-floor to four de-watering trommels, two on each side, the oversize from which is spouted into two 60-in. Steadman disintegrators, operated at 325 rev. per min., where it is crushed to desired size for coke-ovens. The east and west side sections of the jigs are each driven by an 85-h.p. General Electric motor.

The undersize from the trommels is recovered from settling-tanks beneath by perforated-bucket elevators running 15 ft. per min.; and, together with the washed coal from the Stewart and Lührig jigs, is delivered upon conveyor-belt *H*, which carries it to conveyor-belt *G*, the latter traveling a distance of 287 ft. 3 in., to seven 300-ton cylindrical steel storage-tanks, each 20 ft. in diameter, 40 ft. high, and distributed by two drag-conveyors operating above the bins, whence it is taken by electric larries to the coke-ovens. The rejected material from the various washings and re-washings is picked up by elevators and discharged into the waste-tank at the south end of the washery-building, whence it is taken by electric trolley-cars to the waste-dump.

The de-watering trommels, Fig. 8, are driven from the disintegrator line-shaft. The disintegrators are driven by two 200-h.p. General Electric motors. Belt *G*, which conveys the washed coal to the storage-bins, is driven by a 20-h.p. General Electric motor. The two distributing drag-conveyors on top of the washed-coal bins are driven by two General Electric motors, 30 and 20 h.p. respectively. The refuse-elevators are driven by two 5-h.p. Western Electric motors. Two views of the jig-floor are given in Figs. 9 and 10.

The recovery from the oversize from the refuse-trommels carried to Stewart re-wash jigs is a product equal in fuel-value to the unwashed mine-product, and is used as nut-coal for domestic or steam-purposes. This material is carried by belt-conveyor to a circular steel storage-bin.

Twenty-seven electric motors, having an aggregate capacity of 1,159 h.p., are operated in conveying the coal from the tipple and through the crusher-house and washery until delivered in the washed-coal storage-bins. All motors on the alternating current are 3-phase, 25 cycle, 220 volts.

An adjunct common to the mine-tipple of mines Nos. 1 and 2 and to the washery is the "run of mine" crusher situated at the tipple. The crusher is a McCully gyratory No. 7, with a capacity of 200 tons per hour. Should there be any temporary cessation of orders for screened coal for commercial purposes, the whole product of these mines could be crushed and conveyed to the storage-bins to be washed and made into coke.

The washery has proved an eminent success. Even in the experimental stage the fuel-value of the waste was as low as 8 per cent., and the average loss of fuel-values in the waste from the washery now and hereafter will probably be below 5 per cent. The capacity of the plant is 2,500 tons per day of 10 hr., but as there are not a sufficient number of coke-ovens erected to utilize this tonnage, the plant has never exceeded 8 hr. in constant operation. The washery is located in Rail cañon, at a common center to the greatest area of the coal-lands of the company.

A complete laboratory is in a two-story concrete-and-iron fire-proof building, 38 ft. by 26 ft. 6 in., opening into the main washery-building. The lower story is used for grinding and preparing for analysis samples of coal, coke, bone, and waste; the upper story contains the laboratory proper, which is fully equipped with every modern appliance necessary for the work at hand.

All of the machinery for handling the unwashed coal, jigs, and other appliances used in the washing, as well as machinery for handling the washed coal, was manufactured by the Jeffrey Manufacturing Co.

V. Coke-Ovens.

The washed slack is hauled from the storage-tanks to the coke-ovens by two Scott-Dale electric larries, each pulling one trailer. There are 570 coke-ovens in operation: 124 bee-hive ovens, 13 ft. in diameter, and 446 English under-flue ovens, 11 ft. in diameter. Each oven is charged with 6 tons of slack, burns 48 hr., and produces 52 per cent. in weight of coke.

The under-flue ovens are an innovation along economical lines, due to the activity of Dr. Douglas. These ovens are in batteries of from 54 to 58 ovens each, and arranged in a double row, as shown in Fig. 11. The flaming gases from the coke-oven, passing downward into horizontal flues beneath the oven, serve

to coke the slack from the bottom as it is being coked on top, passing thence through an opening in the rear to a main hori-

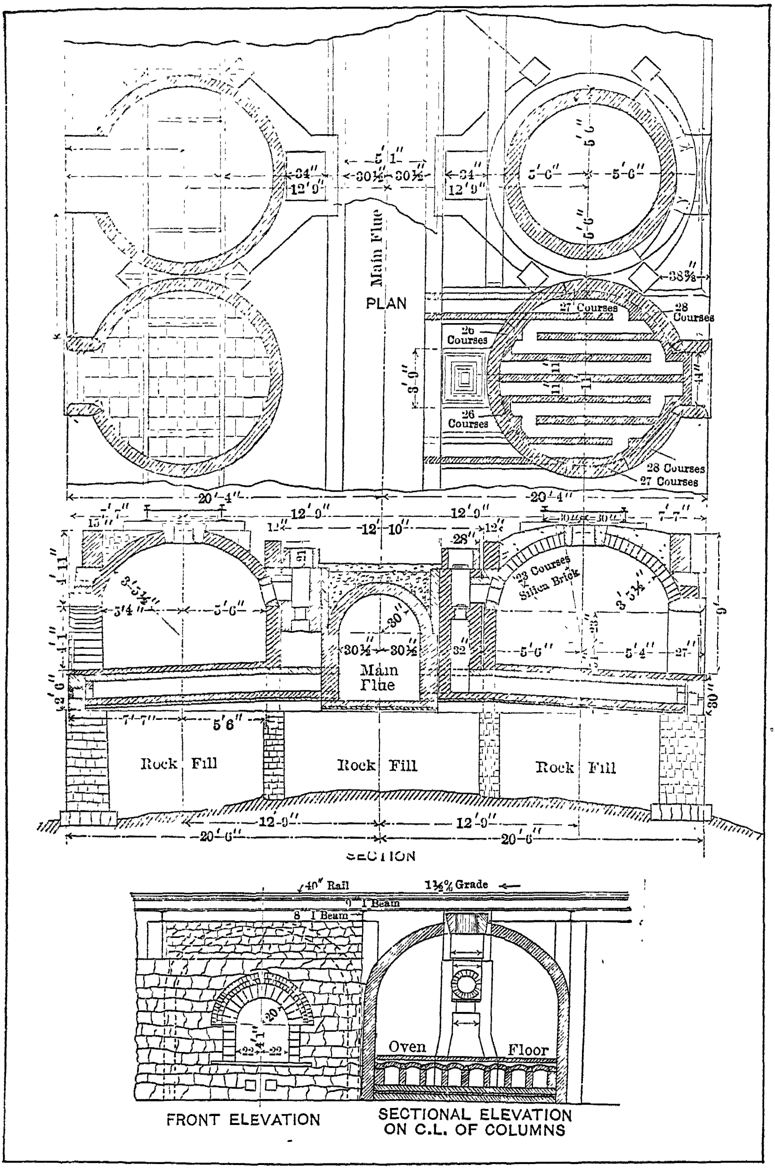


FIG. 12.—PLAN, SECTION, AND ELEVATIONS OF MAIN FLUE AND UNDER-FLUES OF COKE-OVENS.

zontal flue between the two strings of ovens to the boiler-houses, where the heat is used for steam-purposes. The re-

sidual heat and gas pass from the boilers through two brick stacks, 125 ft. in height and 11 ft. in diameter at the top. Details of the construction of these ovens are given in Fig. 12.

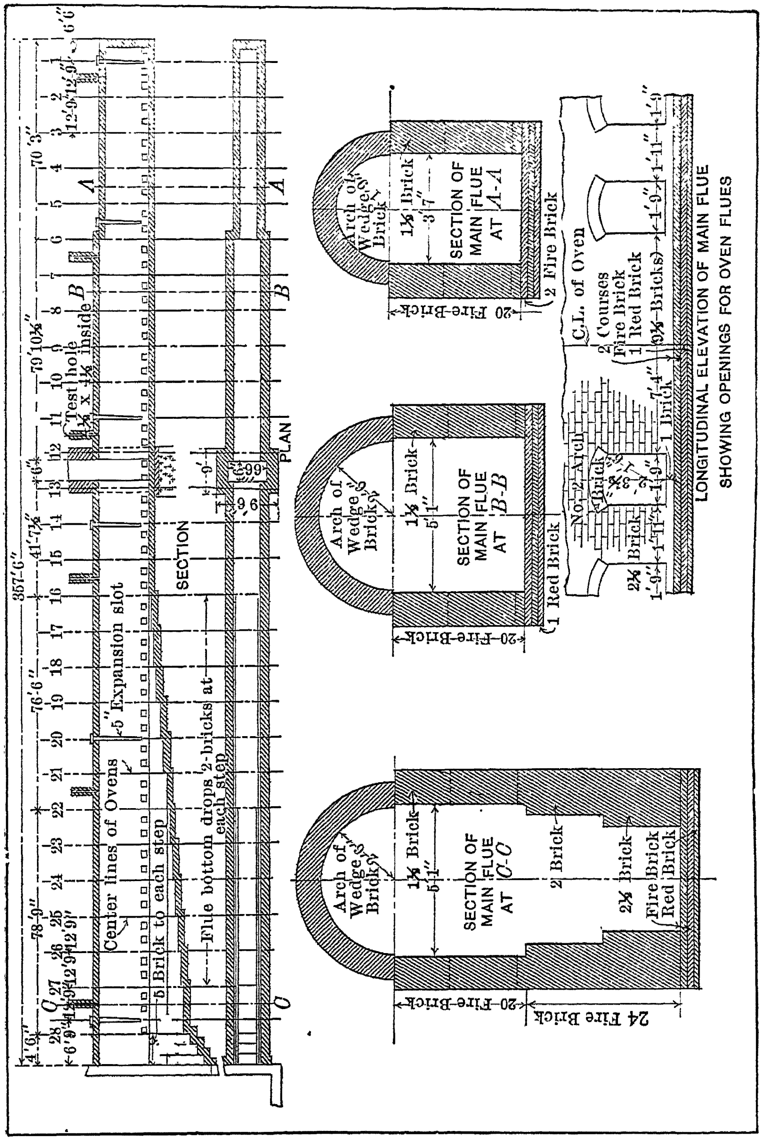


FIG. 13.—ELEVATION, PLAN, AND SECTIONS OF MAIN FLUE FOR BLOCK OF 56 OVENS.

A cross-section of the central flue which conducts the gases from the ovens to the boiler-plant (Fig. 13) has an area of 20.6 sq. ft. at the 27th oven, which is farthest from the boiler-plant

or chimney, and increases as other ovens discharge into it, until at the down-cast to the boiler-plant it has an area of 52.73 sq. feet.

Pyrometer-readings, at the boiler-houses, show that the gases are delivered under the boilers at temperatures varying from 1,800° to 2,600° F., and leave the stack at temperatures of from 600° to 1,150° F.

At present the heated gases from only 218 ovens of the 446 under-flue ovens are being utilized, the return from the other 228 ovens being allowed to pass off through chimneys. Here are vast reserves of power that can be utilized to increase the capacity of the power-plant as the mines increase in extent and production. There is one Covington coke-puller in use at the coke-ovens, electrically driven by two General Electric motors, one of 20 h.p. and the other of 17.5 h.p. It is probable that another coke-puller will soon be in commission.

A good quality of fire-clay has recently been discovered near the coke-ovens; bricks made from it have stood severe tests at high temperatures. A brick-plant has been ordered which will supply all the fire-brick needed for the coke-ovens and other purposes.

VI. Power-Plant.

1. *Boiler-Houses.*—There are two fire-proof boiler-houses situated about 50 ft. apart, on parallel batteries of ovens, the ovens abutting each boiler-plant on both ends. The boiler-houses are identical in construction, having a main room 125 ft. long by 42 ft. wide and 50 ft. high, with brick floors. Everything is clean and quiet, no fuel is in sight, and the temperature is about the same as in an ordinary living-room in a house.

In the east boiler-house there are four Stirling 300-h.p. water-tube boilers, and in the west boiler-house three boilers of similar make and capacity.

The pointers of the steam-gauges on these boilers indicate between 145 and 150 lb. pressure. On opening the front door of the fire-box a dark void is presented, and no heat is radiated from the fire-box. A vagrant ray of light comes from under a narrow sheet of iron about 5 ft. in length on the floor, and moving aside the iron leaves only a thin flooring of brick above the incandescent burning gases beneath. Each boiler is

equipped with a Knowles outside-packed, 7 by 12-in., plunger-pump of a capacity of 275 gal. per minute.

One man attends to both boiler-houses, moving the dampers as necessary to regulate the heat going to the boilers or sending it up the stack as required. In this way both labor and fuel are saved.

The steam is conveyed from the boiler-houses to the power-house through 10-in. steam-lines carried 30 ft. above through structural-iron pipe-galleries.

In addition to furnishing steam for power, the boiler-plant furnishes steam for heating the hospital, theater, amusement-halls, lodge-room, store, office, and other buildings. The steam is taken from the boilers to a sub-station at from 135 to 150 lb. pressure. It is there reduced to from 5 to 20 lb. pressure and distributed as needed to the various buildings.

2. *Power-House.*—The power-house is a fire-proof iron, brick, and concrete structure, 100 ft. long, 50 ft. wide, and 50 ft. high. The plant comprises three cross-compound Nordberg-Corliss engines, long-reach, cut-off type, 19 by 36 by 32 in., direct-coupled to General Electric alternating-current generators, 2,300 volts, 100 amperes, 400 kw. each. The three engines run in parallel. There are two Thompson & Ryan exciters, each 50 kw., 400 amperes, 125 volts, manufactured by Ridgway Dynamo & Engine Co. These exciters magnetize the fields.

The switch-board, of marble, comprises two exciter-panels, three generator-panels, and four feeder-panels, and is equipped with a Terrill voltage-regulator, which keeps the voltage constant with all loads. All the circuits are 3-phase, 25 cycle, on the alternating-current side. A record is made every half hour showing conditions at the power-plant.

The current from the power-house is transmitted by insulated wires at 2,300 volts to rotary converters at sub-stations, where it is converted from 2,300 volts alternating current to 260 volts direct current.

There are three sub-stations, one at Lorita, near mine No. 5, which is equipped with one 200-kw. General Electric rotary converter, 260 volts, 768 amperes. The sub-station at mine No. 4 is equipped with two 200-kw. General Electric rotary converters, 260 volts, 768 amperes. The sub-station between mines Nos. 1 and 2 has an equipment similar to that of mine No. 4.

The electrical-distribution sheet, Fig. 14, shows the motors served by the three generators at the main power-house.

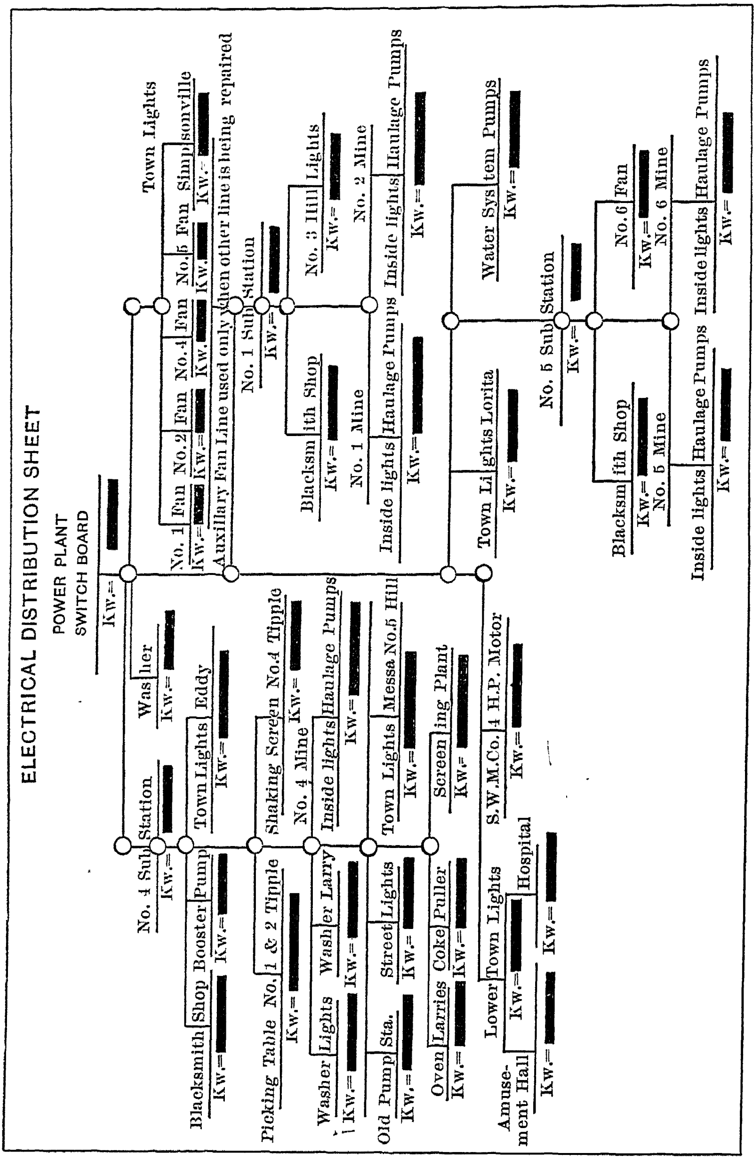


FIG. 14.—ELECTRICAL-DISTRIBUTION SHEET.

The current from each generator is recorded on a watt-meter attached to the switch-board, and from the switch-board six high-tension lines run to various sub-stations, fans, washery,

and town-lighting system, for each line of which a watt-meter is placed at the switch-board. The amount of power used by the various motors is measured by the master mechanic, with a portable watt-meter. The ventilating-fans are served by a high-tension line direct from the switch-board, excepting No. 6, which is served from No. 5 sub-station. These fans are also served by an auxiliary line from No. 1 sub-station, to be used during repairs on the other line, or in other cases of necessity. A reference to the chart shows that the main pumping-station is on the high-tension line with No. 5 sub-station.

The power taken by each of the feed-lines will be the factor used to apportion the expense of power-house and boiler-plant, up to and including the switch-board, among the various operating-accounts. The application of the power, that is, from the switch-board to and including the sub-stations, will be divided, on the basis of power used, among the various accounts served by this line, as shown in the diagram, Fig. 14.

VII. Water-Works.

The water used for domestic and other purposes is taken from a well sunk in the gravels of the river-bottom at a point 3 miles above Dawson, far above any residence, and beyond any opportunity for contamination.

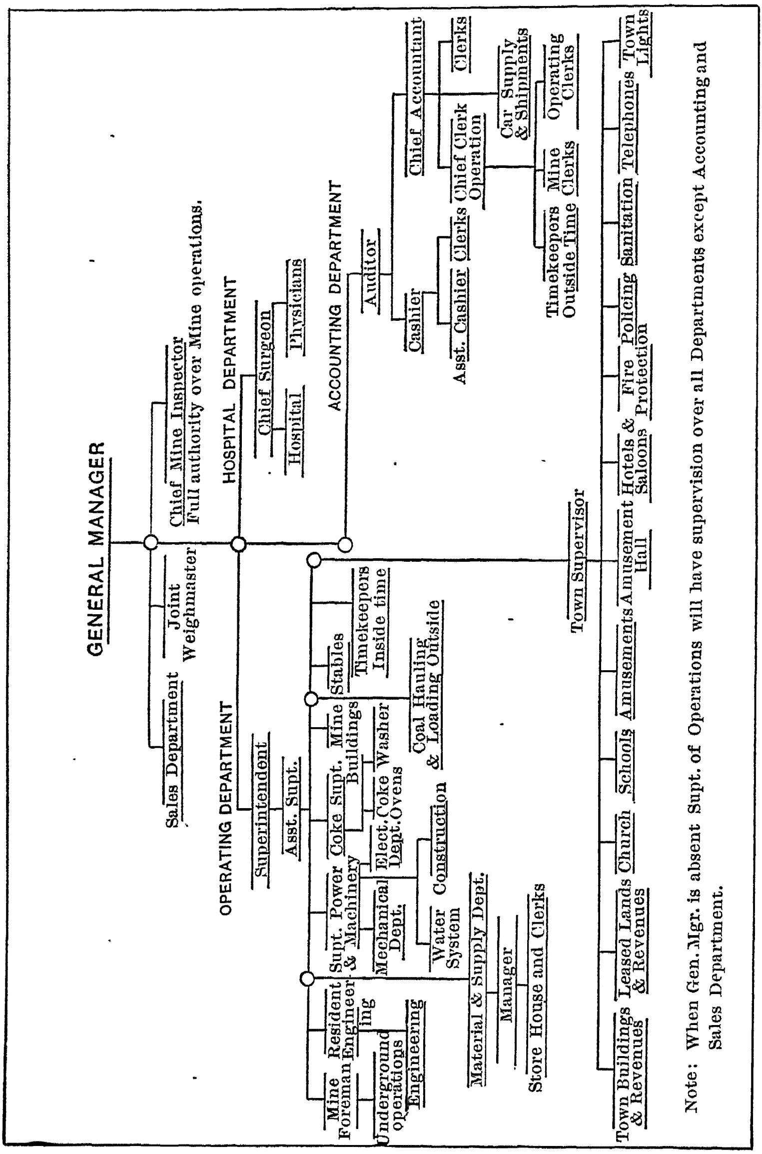
At the main pumping-station are two pumps: one Dean triplex, 11 by 12 in., capacity 596 gal. per min., driven by Western Electric motor, 50-h.p., alternating current, voltage 220, and one Dean triplex, 9 by 12 in., capacity 300 gal. per min., driven by a 30-h.p. General Electric induction-motor, alternating current, voltage 220. These motors are of the squirrel-cage type.

The water is pumped from the well to two 800,000-gal. reservoirs on the hill above the town, at an elevation of about 140 ft. above the houses in camp, whence it is distributed as required.

An auxiliary station is maintained about a mile above the town, on the Vermejo river. This station is kept as a reserve in case of accident to the upper pumping-plant. It is equipped with one Dean triplex, 9- by 12-in. pump, capacity 300 gal. per min., operated by a Westinghouse, 25-h.p., direct-current motor, 260 volts.

In addition to these pumping-stations, there is a "booster" pumping-station at tipples Nos. 1 and 2, which helps to force

water to the houses at greater elevations up Rail cañon, and on the higher *mesas* or table-lands along the cañon. This station is equipped with a Dean triplex, 9- by 12-in. pump, capacity, 300



Note: When Gen. Mgr. is absent Supt. of Operations will have supervision over all Departments except Accounting and Sales Department.

FIG. 15.—ADMINISTRATION-CHART.

gal. per min., driven by a Westinghouse direct-current motor, 25 h.p., 220 volts. This pump is automatically controlled by a rheostat, so that, in case of fire, the pump could be speeded

up and used as a fire-pump, and to keep the water-supply replenished.

VIII. Administration.

"System" is the watchword of successful business venture, and every effort is being made towards perfection in all the departments of the extensive business of the company. The chart, Fig. 15, shows the various branches of the administration, together with the respective degrees of authority.

IX. The Civic Features of Dawson.

1. *General.*—The town of Dawson, with its suburbs, has a population of 4,000, of which 1,600 adults are employed in and about the mines, coke-ovens, coal-washery, etc., in addition to the men employed on the railroad, in hauling timbers, etc.

There are 594 houses, each containing from four to eight rooms, including some larger domiciles for boarding- and lodging-houses. The houses are of various designs, situated in valleys and on hillsides, and producing a pleasing scenic effect. They are well supplied with pure water from a clear mountain stream, the Vermejo river, and lighted by a good electric-light system. House-rent is at the rate of $2 per room, about one-half of the usual rent for similar houses in other towns and cities outside of coal-camps.

Electric lights cost 25 cents per month for each 16-c-p. light, and 50 cents for 32-c-p. lights. This also is one-half the price charged in other towns and cities in New Mexico. Water is free.

Each employee pays $1.50 per month for medical attendance for himself and family, if he has a family. This charge covers medicines, admission to the hospital, and surgical operation, when necessary. The hospital is modern in every particular, and its facilities are far superior to those of most towns and cities of similar size. Three first-class physicians and several skilled nurses are employed in the hospital. An ambulance of modern design is always available, and saddle-horses are at hand for the use of the physicians in responding to emergency-calls.

2. *Amusements.*—The company has built a large theater and amusement-hall, in the basement of which are bowling-alleys open to ladies and gentlemen. On the first floor is a beautiful

theater; on the same floor at the side of the theater is a large billiard-parlor. On the second floor are the galleries of the theater, and a large and well-furnished lodge-room, where the various societies hold their regular meetings. The theater-building cost the company about $35,000. Only a nominal charge is made for the use of the amusement-halls and lodge-rooms. Generous inducements are offered to theatrical companies to present plays.

3. *Churches.*—There is a large and commodious church, heated by a furnace, both fuel and light being provided by the company, free of charge. An Episcopal clergyman is in charge of the pastorate, but the church is open to all denominations wishing to hold religious services.

4. *Schools.*—Two large school-houses have been built, one at the expense of the school-district and one by the company. A smaller building belonging to the company is also used for school-purposes at No. 5 mine. The company collects, in accordance with the Territorial law, an annual tax of $1 from each employee. The money is given to the county school-fund, and the proportion belonging to the Dawson school-district is returned to the school-trustees of the district. The estimated cost of maintaining the Dawson schools during the ensuing year is $12,000, of which the county school-fund will appropriate $5,000; the company has already appropriated $6,000 to make up the deficiency, and will probably be called upon to appropriate $1,000 in addition.

The Dawson schools are the only ones in New Mexico in which a full 10 months' scholastic term is held. Nine teachers and two janitors are employed, and the total enrollment of children of school age is 445, of which the average daily attendance is 338. A high school and a kindergarten will be added within the next year.

5. *The Store.*—The company maintains a store, supplying all the necessities and many of the luxuries of life at prices which compare favorably with those charged in other towns and cities of the Territory. The prices of food-products are lower than those which prevail outside the coal-camps.

6. *The Bank.*—The bank is one of the prominent factors in the welfare of the employees, many of whom deposit their earnings from time to time, receiving interest thereon at the rate

of 3.5 per cent. per annum, compounded semi-annually. This provision teaches thrift and induces economy, to the betterment of the laboring-man. Bills of exchange are issued upon all the principal cities of the United States and Europe.

X. Remarks.

I desire to say a few words of deserved tribute to the men at the helm of this great enterprise. These men may be justly called "industrial optimists"; with an optimism based upon confidence in their own good judgment, which in turn is based upon careful and intelligent investigation of the property offered for development.

The Dawson coal-field is only one of many great industrial enterprises in the West which these men have developed and brought to a state of successful fruition. Having once determined that a property has legitimate merit as a business undertaking, without hesitancy they supply the necessary capital to develop its possibilities, and give personal attention and efforts to make it a success, and their solicitude for the welfare of their employees has become axiomatic. They have for many years past been one of the most potent factors in the progress and upbuilding of the Southwest.

In closing, I wish to acknowledge publicly my obligation to the following men connected with the Stag Cañon Fuel Co., who have so kindly and willingly furnished the information concerning the details of working-plans and the operating of the mines:

To Dr. James Douglas, for *carte blanche* to have access to all office data, as well as plans and drawings; to E. L. Carpenter, general manager, for similar concessions; to Messrs. David Crow, general superintendent; F. N. Cameron, chief mine-inspector for the company; F. H. Weitzel, chief engineer; T. H. O'Brien, builder of the washery; Roy Mounday, master mechanic; Thad. Kinney, T. W. Lewis, G. M. Hanson, as well as other employees, for uniform courtesy in imparting desired information, not alone in any one instance, but in the general performance of my official duties.

Pan-Amalgamation: an Instructive Laboratory-Experiment.

BY H. O. HOFMAN AND C. R. HAYWARD, MASSACHUSETTS INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(New Haven Meeting, February, 1909.)

I. Introduction.

The aim of instruction in a metallurgical laboratory is to make real the principles on which metallurgical processes and operations are based, and to foster the spirit of investigation. The materials with which experiments are carried out are ores, metals, and metallic compounds. The method varies with the end sought. A class may work as a whole, each member contributing his share to the solution of a problem, or the students may carry on investigations independently; the former exemplifies class-research; the latter, individual research. It is with a branch of the former, with special reference to ore-treatment, that the present paper deals.

In smelting an ore by a well-established process, the result is shown by analyzing the products to see whether their compositions correspond to those calculated in making up the charge; by taking account of stock to show the distribution of metal in the different products made and the losses from dust and volatilization; by casting a thermal balance to find the distribution and losses of heat; and by making a cost-sheet to ascertain, as far as possible, the necessary outlay of money.

In lixiviation and amalgamation, the mode of operating has to be varied to adapt a process to the individual ore. Here a number of tests become necessary. Each will consist of a series of experiments with one variable, in order to find the conditions under which the variable gives the best result; a summary of several tests will give the best method of operating. Working a number of charges with the best method will furnish the data desired for ascertaining the recovery of metal,

the size of plant needed for a given capacity, and the cost of treatment.

The Washoe process, raw-amalgamation of a silver-ore in an iron pan, furnishes a satisfactory example of this class of ore-treatment. The object of the present paper is to show how pan-amalgamation is carried on as a class-exercise at the Massachusetts Institute of Technology.

II. The Amalgamating-Pan.

The first amalgamating-pan with settler was put in operation at the Massachusetts Institute of Technology in 1871.[1] It was built on the Washoe pattern by Messrs. Booth & Co., San Francisco, Cal. In 1895 the laboratory had three of these pans,[2] respectively 30, 18, and 12 in. in diameter. They were, however, little used at that date for class-work on account of the time required to get them into good working-order, and of the difficulties met with in making a clean-up, as it was next to impossible to recover all the amalgam from a pan with detachable shoes and dies. While the percentage of extraction is usually based upon the assay of the tailings, it is of importance in a teaching-experiment to compare it with the actual yield from the amalgam. Instruction was given mainly with small pans, only 7 in. in diameter, three of which were of copper, hardened by a small percentage of silicon, and the rest of cast-iron. A drawing and a brief description of these have already been given in a paper read before the Institute.[3] These pans were a great advance over the original Washoe model, but improvements in the details of construction suggested themselves, which led, in 1899, to the replacement of the pan of 1895 by the present form. This has met all the requirements of a pan that is to be used for class-work in the systematic testing of ores. These requirements are that it shall give a quantitative result which corresponds to working-conditions, and that it shall be small, easy to run, and easy to clean.

The battery of ten pans in the laboratory is represented in Fig. 1, and detailed drawings of the pan in Figs. 2 and 3. The pan, Fig. 2, is cast in one piece. It has a flat bottom, which

[1] Richards, *Trans.*, i., 400 (1871–73).

[2] Hofman, *Trans.*, xxv., 326 (1895).

[3] *Loc. cit.*

forms the lower grinding-surface; its inside dimensions are: diameter at bottom, 7 in.; at top, 7.25 in.; height, $4\frac{5}{8}$ in. In the center is a hollow core, 2.75 in. in diameter and 3.75 in. high, to prevent the pulp from collecting. The pan has four legs, which stand on a wooden stool; the latter carries a flat evaporating gas-burner for heating (not shown in the illustration).

The muller, Figs. 2 and 3, is of special construction. It is cast in one piece, as is the pan. The upper part, the driver, is slipped over the rotating-shaft and fastened to it by a set-screw; it has a vent, Fig. 2, to prevent hot pulp from being sucked into the core; the spider has two legs only; the form of the muller-plate and shoes is given in Fig. 3; details of construction are given in Fig. 2. The shoes have the usual form of an oblique sector of a circle. One peculiarity of the shoe, seen in half-section on *D E* and in section on *L M* of Fig. 2, and in Fig. 3, is that the part outside of the muller-plate tapers from 0.25 in. at the front to $\frac{7}{8}$ in. at the rear end, and thus assists in the formation of a pulp-current by raising the pulp while the muller is being rotated through the driving-shaft.

This shaft, Fig. 2, is suspended from a bevel-gear with hub, fastened to it by a set-screw, and journaled in two boxes bolted to a wooden frame common to the ten pans, Fig. 1. The bevel-wheel is driven by an adjustable bevel-pinion, which is thrown in and out of gear by a forked lever (stopper), the arms of which end in a grooved hub; the lever is supplied near the bottom with a hook, to be lowered into an eye (not shown) when the pinion is working.

III. Mode of Operating.

During the winter term the whole of Tuesday and the afternoon of Wednesday are given to class laboratory-work; and the following Saturday one hour is devoted to the discussion of results. Exercises in pan-amalgamation are so planned that a student starts his experiment Tuesday morning and finishes it Wednesday afternoon; the several results are handed to an instructor, who tabulates (Table I.) and plots them (Fig. 4), and prepares manifold sheets, which are distributed at the conference on the following Saturday, when the work as a whole is passed in review. Thus a series of as many as 10 tests, with

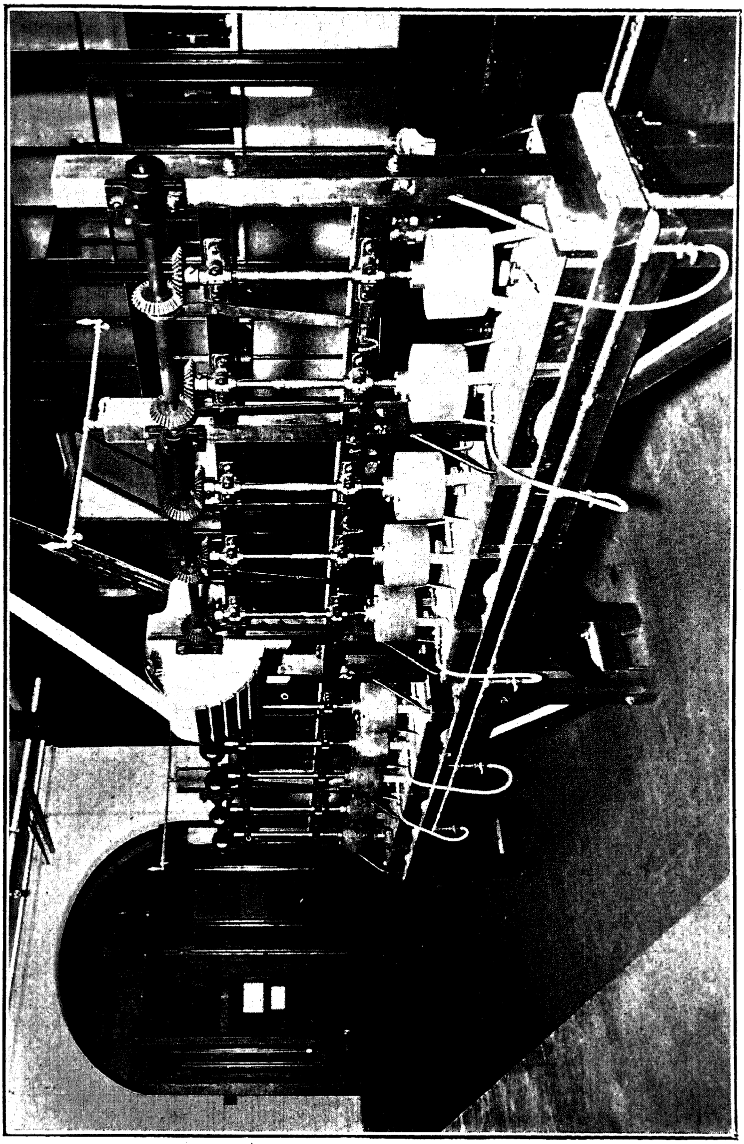


FIG. 1.—BATTERY OF AMALGAMATING-PANS.

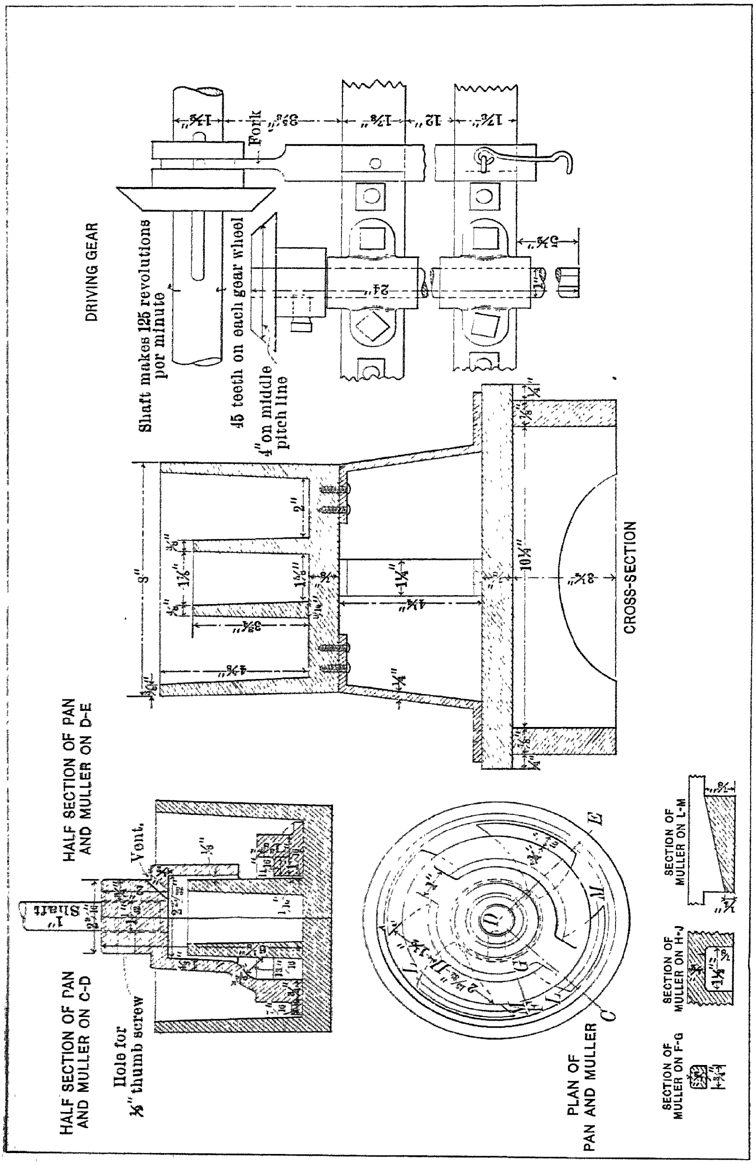


FIG. 2.—THE RICHARDS LABORATORY AMALGAMATING-PAN.

one variable—for example, the time of grinding—is carried out by the class in one and one-half days, and the whole experiment finished up the same week. Another section of students will make a test with the same ore, choosing a different variable—for example, the time of amalgamating—and carry it through in a similar way. In this manner the principal variables in pan-amalgamation are taken up by class-sections, each section benefiting by the work of the others, while at the same time the best manner of treating the ore under consideration is being investigated.

The details of an operation may be given in connection with Table I. and Fig. 4, which represent a series of 10 tests, in which

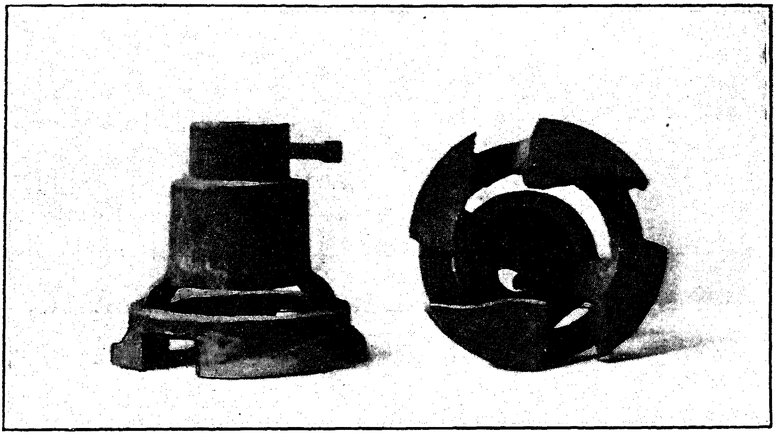


FIG. 3.—MULLER.

the time of grinding was the variable, ranging from 20 to 100 minutes.

The pan is first cleaned. For this purpose the muller is raised on the shaft and clamped, the wooden stool under the pan is withdrawn, the pan taken out and dusted, and the suspended muller freed from adhering particles of foreign matter. The pan is now put in place, the muller lowered, pressed down, and turned to and fro by hand and clamped. The necessary amount of water, 500 c.c., is charged, the muller set going, the lamp lighted, and the salt, 180 g., added. The ore, 1,800 g., of 40-mesh size material, is fed in slowly, and the time of grinding counted after all the ore has been charged—namely, after about 5 min. On account of heating the pulp with a lamp to about

80° C., there is considerable evaporation of water during the grinding- and amalgamating-periods, which has to be remedied by adding fresh water, in this case from a wash-bottle holding 500 c.c. The amount used is noted, as it gives an idea of the care with which the heating has been carried on. Allowing the pulp to become too thick requires an excess of water over the normal to thin it down in order that the desired current may be again established. Table I. shows that the water-additions ranged from 465 to 869 c.c. Water from the wash-bottle should be blown in small amounts against the side of the pan; it will

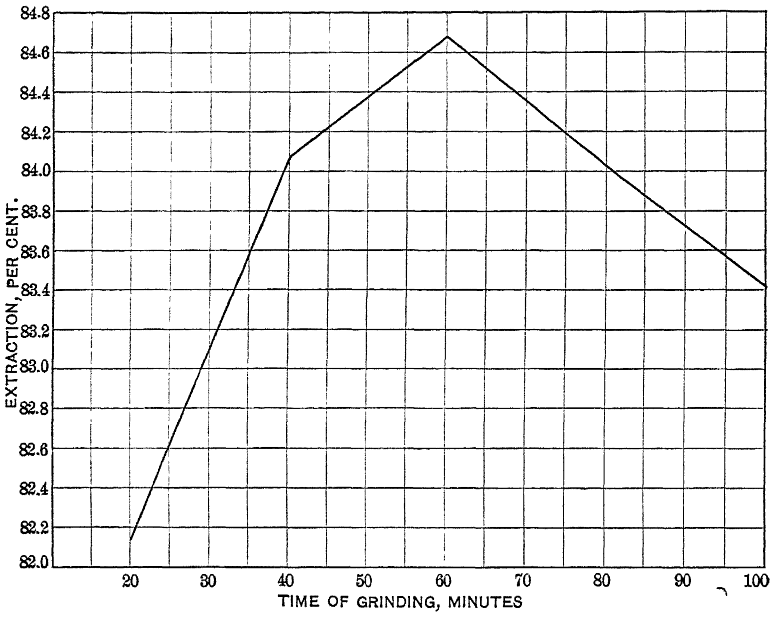


FIG. 4.—AMALGAMATION WITH TIME OF GRINDING AS VARIABLE.

loosen parts of the top of the charge which have adhered to the warm pan and become hard; the pulp-current then will carry them towards the center and cause them to descend there. Scraping the sides with a spatula corrects the adhesion of parts of the charge, and has to be resorted to more or less during the larger part of a test, as repeated additions of water thin the pulp to such an extent as to spoil the current.

At the end of the grinding-period, the muller is raised $\frac{1}{8}$ in. previous to adding the quicksilver. In order to fix the distance, a pencil is held against the rotating shaft and a line

TABLE I.—*Details of Amalgamation with Time of Grinding as Variable.*

Name.	Number of Pan (Iron).	Weight of Ore.	Water Used.			Weight of Salt.	Weight of Mercury.	Time.		Tailings.							Weight of Amalgam Recovered.	Weight of Bullion from Retort.	Weight of Mercury Recovered.	Silver Account.					Silver Amalgamated.	
			At Start.	During Run.				Grinding.	Amalgamating.	On Filter.		Through Filter.		Lost.		Assay, Ag.				Weight in Ore.	Weight in Tailings, Based on 1,830 Grams.	Weight in Amalgam.	Total Weight.	Amount Accounted for.	Based on Assay of Tailings.	Based on Silver in Amalgam.
				Quantity.	Per Cent. of Total Water.					Weight.	Amount.	Weight.	Amount.	Weight.	Amount.											
		Gm.	C. C.	C. C.		Gm.	Gm.	Min.	Hr.	Gm.	Per Cent.	Gm	Per Cent	Gm	Per Cent.	Ounces per Ton.	Gm.	Gm	Gm	Gram.	Gram.	Gram.	Gram.	Per Cent.	Per Cent.	Per Cent.
A	1	1,800	500	755	60.16	180	150	20	$2\frac{1}{2}$	1,797	99.84	2	0.11	1	0.06	15.59	159	10	1,487 grams = 99.13 per cent.	5.499	0.962	4.516	5.478	99.61	82.51	82.13
B	2	1,800	500	675	57.44	180	150	20	$2\frac{1}{2}$	1,775	98.60	6	0.33	19	1.07	16.68	162	12		5.499	1.029	4.470	5.380	99.83	81.74	81.28
C	3	1,800	500	849	62.94	180	150	40	$2\frac{1}{2}$	1,780	98.88	7	0.38	13	0.74	14.15	158	12		5.499	0.873	4.618	5.491	99.86	84.10	83.97
D	4	1,800	500	750	60.00	180	150	40	$2\frac{1}{2}$	1,785	99.15	5	0.28	10	0.57	14.22	164	10		5.499	0.878	4.608	5.486	99.77	84.03	83.79
E	5	1,800	500	800	61.55	180	150	60	$2\frac{1}{2}$	1,778	98.78	2	0.11	20	1.11	14.00	161	11		5.499	0.844	4.654	5.498	99.97	84.58	84.65
F	6	1,800	500	630	55.75	180	150	60	$2\frac{1}{2}$	1,780	98.88	3	0.16	17	0.96	13.92	163	11		5.499	0.859	4.624	5.483	99.70	84.76	84.09
G	7	1,800	500	550	52.38	180	150	80	$2\frac{1}{2}$	1,789	99.38	8	0.44	3	0.16	14.30	158	13		5.499	0.883	4.688	5.471	99.49	83.95	85.23
H	8	1,800	500	500	50.00	180	150	80	$2\frac{1}{2}$	1,792	99.56	1	0.06	7	0.38	14.16	160	11		5.499	0.874	4.608	5.482	99.70	84.09	84.09
I	9	1,800	500	465	....	180	150	100	$2\frac{1}{2}$	1,778	98.78	5	0.28	17	0.96	14.90	156	12		5.499	0.920	4.570	5.490	99.84	83.28	83.12
J	10	1,800	500	610	54.95	180	150	100	$2\frac{1}{2}$	1,785	99.15	7	0.38	8	0.47	14.64	163	10		5.499	0.904	4.579	5.483	99.70	83.56	83.28

marked off; the bevel-wheel is now thrown out of gear, the muller unclamped, raised, reclamped, and set rotating again. A weighed amount of quicksilver, 150 g., is then added during about 5 min. in a fine spray from a glass funnel on to the charge near the outer edge. The consistency of the pulp must be a little thicker than during grinding; the right degree will be determined by the manner in which the quicksilver is disseminated; this will be found in globules if the pulp is too thick, in fine particles if right, and will collect on the bottom if too thin.

In making a clean-up, the first step is to remove the tailings and amalgam from the pan. The muller is stopped; a sheet-iron vessel, 18 in. square and 4 in. deep, having a thin coat of pitch and tar, is placed underneath the pan; the muller is again set going, and first 1 liter of water added to thin the pulp, then 2 liters more in about 5 min., which causes a large part of the tailings to overflow into the vessel. The amount of water desired and the time allowed for adding it had been settled by experiment before adopting this mode of operation. The rest of the pan-content is now transferred to a fiber pail holding about 2.5 gal., the pan and muller being scraped with a spatula and brushed with a dauber. The next step is the separation of the amalgam from the tailings and the recovery of the latter. The tailings collected in the iron vessel are transferred to a filter, which is a simple wooden frame, 18 in. square and of 1-in. section, with heavy unbleached cotton cloth spread over it and nailed fast on the under side. The 10 filters are soaked for several hours in water before they are put to use, in order to close the pores. Nevertheless, small amounts of slime pass through, which are caught with the filtrates in buckets and allowed to settle over night, when the clear liquid is decanted and the slime collected from each filter, dried and weighed ("through filter" in Table I.). The tailings and amalgam, collected in a bucket, are separated by panning twice in a 16-in. gold-pan; the tailings go on to the filter-cloth and drain over night; the amalgam, collected in a porcelain dish, is dried and weighed. The discrepancy in weights of quicksilver and amalgam in the table requires explanation. The combined weights of quicksilver fed, 150 g., and silver contained in the charge, 5.499 g., give 155.499 g.,

while the weights of the amalgams recovered show a range of from 156 to 164 g. Part of this excess is due to a possible slight overweight in the quicksilver charged; part, however, to the presence of impurities in the amalgam. Thus, a partial analysis of retort-bullion gave: Ag, 51; Pb, 43.61; Fe, 5.12; Cu, tr.; total, 99.73 per cent. The 10 amalgams of a series of tests are placed separately in half-cylinder cast-iron vessels, transferred to a pair of retorts, and distilled in a two-muffle furnace, which is fired with soft coal; the muffles are 4 in. wide, 6 in. high, and 18.25 in. deep. Each vessel is coated with chalk, and

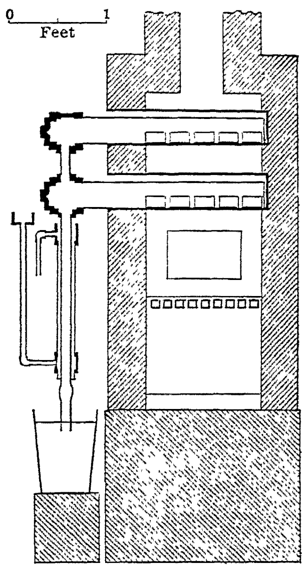


FIG. 5.—AMALGAM-RETORT.

receives a layer of paper before the amalgam is charged, in order to prevent the retort-bullion from adhering to the iron.

The general arrangment of apparatus is clear from Fig. 5. It consists of two wrought-iron pipes, 3 in. in diameter and 24 in. long, each closed at one end by a disk, 0.75 in. thick, that has been welded in, and at the other by a reducing T, 3 by 3 by 1 in., and a reducing cross, 3 by 3 by 1 in., respectively, and joined by a 1-in. connecting-pipe; the T and the cross are closed by square-head screw-plugs. Into the lower retort is screwed the condenser, which reaches into a vessel filled with water.

The quicksilver is driven off in about 3 hr.; the water in

the condenser has to be replaced at intervals; a continuous flow of water was found to be unnecessary. The retorts and the furnace are allowed to cool over night.

The morning after the run has been made, the drained filters are placed on steam-tables to become thoroughly dry, and the retort is opened. In the afternoon, each student receives the tailings and the retort-bullion from his pan; he passes the tailings through a 40-mesh sieve to break up the lumps, samples them down to 200 g., crushes the sample through a 100-mesh sieve, and makes a duplicate assay; at the same time, he scorifies his retort-bullion and cupels it. By using a large muffle, 8.5 by 5 in. and 18 in. deep, a number of crucible-fusions can be made at the same time, and thus the work expedited. The results are handed to the instructor, who records the weights and assays.

In Table I., the weights of tailings on the filters range from 1,775 to 1,797 g., equal to from 98.60 to 99.84 per cent.; those that passed through the filters weigh from 1 to 8 g., equal to from 0.06 to 0.44 per cent., which gives a loss in weight ranging from 1 to 20 g., or from 0.06 to 1.11 per cent. The weights of amalgam and retort-silver show some variations. The combined recovery in quicksilver from the 10 tests is high, 99.13 per cent. In making up the silver-account, the tailings show assay-values of from 13.92 to 16.68 oz. of silver per ton. A pan was charged with 5.499 g. of silver; this is the total to be found in the tailings and in the amalgam; the amount accounted for is seen to vary from 99.49 to 99.97 per cent. The last two columns give the extraction in silver based upon the tailings-assay and upon the recovery in the amalgam. The former figure is, of course, the only reliable one, but the other column is added to bring out any contrasts which may exist, as they form a valuable means for instruction. It is an accident that the figures of the two columns agree so closely; frequently considerable discrepancies occur, due to imperfect cleaning of the pan in a preceding test, or to hard amalgam adhering to the muller in one case or peeling off in another.

Fig. 4, finally, shows graphically the extraction of silver as influenced by the time of grinding. It is seen to increase with the time of grinding from 20 to 60 min., when it reaches a maximum of 84.76 per cent., and then to fall off. The prob-

able reason for the diminished yield, after 60 min. of grinding, is the excessive sliming of the ore, which affects harmfully the pulp-current and flours the mercury, which increases the losses. Without a good current a satisfactory extraction is hardly ever obtained.

IV. Summary of Tests.

The following is a summary of a large number of tests made in extracting the silver from a single ore by raw-amalgamation in cast-iron pans of the construction given. They represent the first experiences of students in this kind of work, who, however, are familiar with assaying and panning. In the selection of samples only those tests have been omitted which in the class-conferences were decreed to be faulty for some well-ascertained reason.

The ore is a silver-ore from the Palmarito Mining Co., District of Mocorito, Sinaloa, Mexico. An examination, aided by the microscope, showed that it was composed mainly of quartz and kaolinite, and contained besides some hematite, galena, pyrite, native silver, and cerargyrite. In the pulp, crushed by means of rolls through a 40-mesh sieve, were found particles of metallic iron. The ultimate analysis gave: H_2O, hygr., 0.07; SiO_2, 86.10; Fe, 6.68; Al_2O_3, 2.66; S, 0.07; Pb, 0.28; Ag, 0.31 (89.1 oz. per ton); As, Sb, Cu, absent.

The rational analysis was determined by the following considerations: The Al_2O_3 was calculated as kaolinite (Al_2O_3, 39.8; SiO_2, 46.3; H_2O, 14.9); the remaining SiO_2 was assumed to be quartz; the S not required by Pb to form galena was calculated as being bound to Fe as pyrite; metallic iron to the extent of 0.10 per cent. was extracted by a magnet from the pulp; the remaining Fe was figured as hematite; of the Ag present, 0.02 per cent. was extracted by means of sodium hyposulphite and calculated as cerargyrite; the rest was assumed to be present in the metallic state. This procedure was believed to be warranted by the facts that As and Sb were absent, that more than 70 per cent. of the total silver was amalgamated in 20 min., and that more than 80 per cent. was recovered in the pan in the absence of salt. The rational analysis of the pulp thus gave: H_2O, hygr., 0.07; quartz, 83.01; kaolinite, 6.68; hematite, 9.36; galena, 0.38; pyrite, 0.06; metallic iron, 0.10; cerargyrite, 0.03; metallic silver, 0.29; total, 99.98 per cent.

In the tests there were examined the effects of varying the addition of salt, the time of grinding, the time of amalgamating, and, last, the influence of an addition of blue vitriol. Previous experiments had shown that, with a charge of 1,800 g. of ore, 500 c.c. of water at the start gave a satisfactory pulp, and 150 g. of quicksilver an amalgam of sufficient liquidity to reduce the loss in panning to a negligible quantity. These three items, therefore, were kept constant in all the work, as well as the temperature, which was held at about 80° C.

TABLE II.—*(Series I.) Effect of Varying the Amount of Salt.*
(Time of grinding, 1.5 hr.; time of amalgamating, 2 hr.).

Weight of Salt Added.		Extraction of Silver.	Number of Tests on which Extraction is Based.
Grams.	Per Cent.	Per Cent.	
....	....	80.83	3
54	3	81.35	4
108	6	85.16	6
180	10	83.97	15
270	15	85.19	6

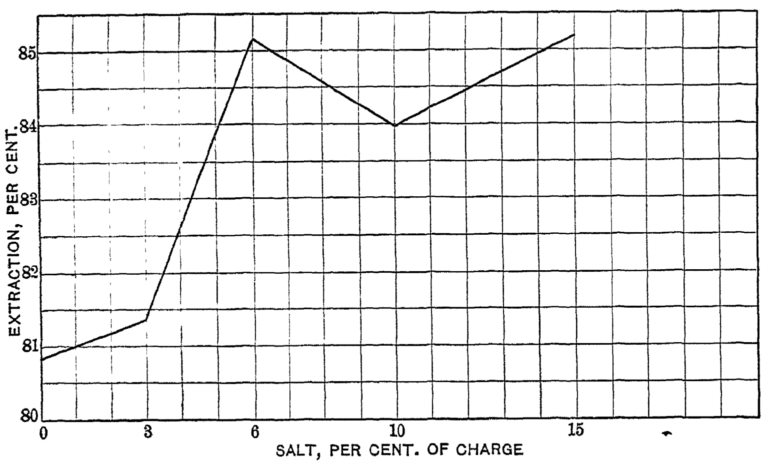


FIG. 6.—SERIES I.: EFFECT OF VARYING THE AMOUNT OF SALT.

The results, recorded in Table II. and Fig. 6, show that, while the extraction in the absence of salt is high, it rises rapidly until 6 per cent. of salt has been added, falls with 10 per cent., and rises again to practically the former maximum when 15 per cent. of salt has been charged.

(The reason for the falling-off in extraction between 6 and 15 per cent. of salt is not clear, and will have to be looked into at a later date. The salt-series was the last one that was investigated; in the other tests the usual standard addition of 10 per cent. had been made; this explains the discrepancy between advocating 6 per cent. of salt and using 10 per cent.)

There is, therefore, no reason for going beyond 6 per cent. The high extraction without the presence of any salt whatever points to the supposition that a large part of the silver is present in the metallic state.

TABLE III.—*(Series II.) Effect of Varying the Time of Grinding.*

(Time of amalgamating, 2.5 hr.; weight of salt added, 180 g.)

Time of Grinding.	Extraction of Silver.	Number of Tests on which Extraction is Based.
Min.	Per Cent.	
....	79.82	6
20	82.21	8
40	84.21	4
60	85.31	2
80	84.75	4
90	84.11	30
100	83.11	4

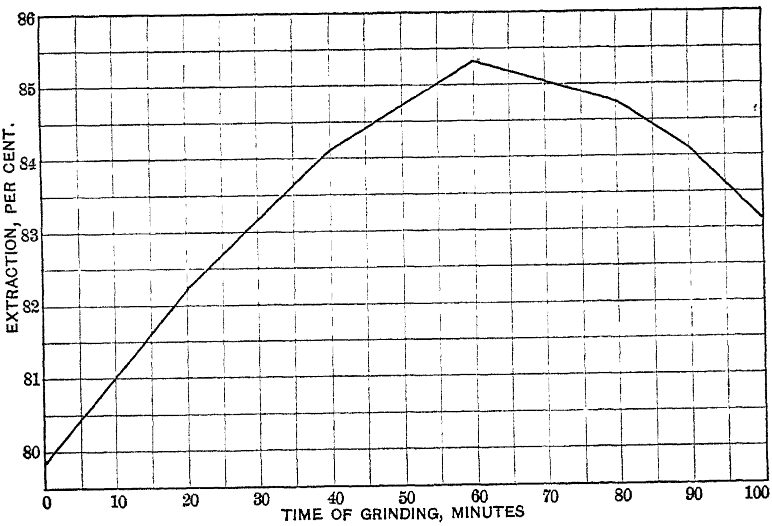


FIG. 7.—SERIES II.: EFFECT OF VARYING THE TIME OF GRINDING.

The extraction, Table III. and Fig. 7, is seen to resemble very closely that given in Table I. and Fig. 4.

TABLE IV.—(*Series III.*) *Effect of Varying the Time of Amalgamating.*

(Time of grinding, 1.5 hr.; weight of salt added, 180 g.)

Time of Amalgamating.	Extraction of Silver.	Number of Tests on which Extraction is Based.
Hr.	Per Cent.	
0.5	74.02	4
1	84.14	4
1.5	84.66	6
2	83.97	14
2.5	84.11	30
3	84.19	11
3.5	84.36	11
4	83.71	13

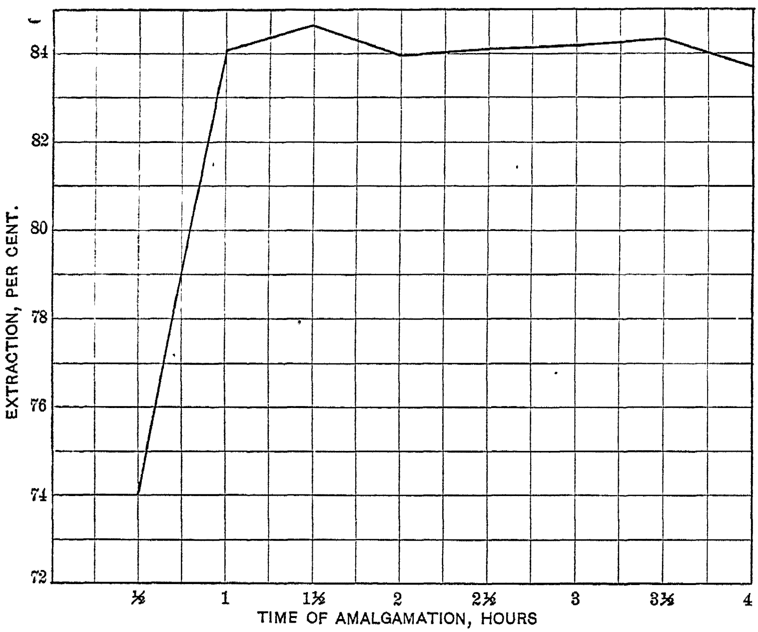


FIG. 8.—SERIES III.: EFFECT OF VARYING THE TIME OF AMALGAMATING.

The data, Table IV. and Fig. 8, show clearly the rapid rise in the extraction during the first hour of amalgamation, and the small increase during the next half-hour, when a maximum is reached. The slight falling-off later on is to be attributed to the inevitable flouring of quicksilver in every amalgamation-process, with a consequent loss in silver.

TABLE V.—*(Series IV.) Effect of Adding Blue Vitriol.*

(Time of grinding, 1.5 hr.; time of amalgamating, 3 hr.; weight of salt added, 180 g.)

Weight of Blue Vitriol Added.		Extraction of Silver.	Number of Tests on which Extraction is Based.
Grams.	Per Cent.	Per Cent.	
....	....	84.19	11
1.5	0.08	83.03	3
2	0.11	83.68	2
2.5	0.14	81.74	2
3	0.17	82.21	4
3.5	0.19	80.25	2
6	0.33	79.30	2
8	0.44	79.55	2
10	0.55	82.25	2

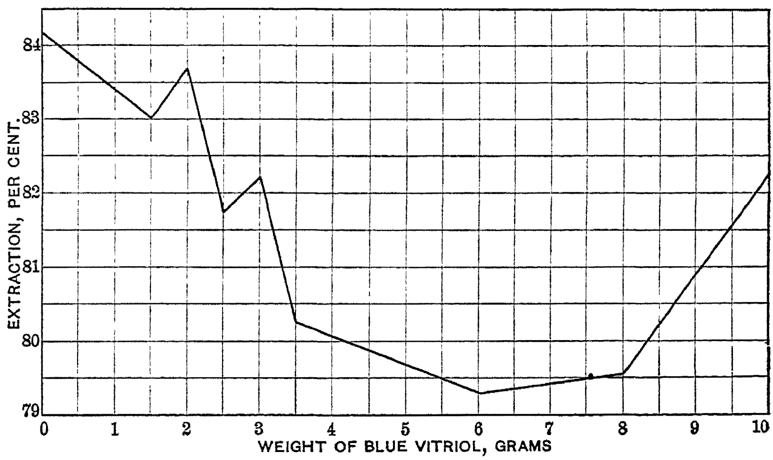


FIG. 9.—SERIES IV.: EFFECT OF ADDING BLUE VITRIOL.

The addition of blue vitriol to the pan, as shown in Table V. and Fig. 9, has no beneficial effect whatever; on the contrary, the extraction decreases. The irregularities in the results are due to the amalgamation of copper, which causes losses in panning and in the subsequent assaying.

The inferences to be drawn as to the treatment of the ore are that, with a charge of 1,800 g., with an addition of 500 c.c. of water at the start and smaller amounts later on to keep the consistency of the pulp constant, and with 150 g. of quicksilver, 6 per cent. of salt, 60 min. grinding and 90 min. àmalgamating give the highest extraction.

V. CONCLUSION.

The data and the curves drawn from them show that the results are satisfactory, and especially so when it is remembered that they represent the work of students making such tests for the first time. Considering pan-amalgamation as a laboratory-experiment, it teaches in a simple, quick, and effective way, with an apparatus that is inexpensive, the importance of series-work in making an investigation and the value of taking account of stock; it further gives results that can be used as the basis for work on a large scale.

Pressure-Fans vs. Exhaust-Fans.

BY AUDLEY H. STOW, MAYBEURY, W. VA.

(New Haven Meeting, February, 1909.)

I. INTRODUCTION.

THE general drift of the discussion as to the relative merits of pressure- and exhaust-fans has resulted, if we may judge from the prevailing practice, largely in favor of the latter. The subject has been hitherto treated, however, with special reference to highly gaseous mines, the effect of the form of ventilation upon dust-explosions having been too little considered. Even in mines containing practically no gas, dust-explosions may occur; and if, as I believe, pressure-fans are to be preferred for their effect upon dust, it may be worth while to reopen the debate, and to challenge the advisability of exhaust-ventilation under certain conditions.

There are several forms of colliery-ventilation, which may be described as follows:

Plain or simple pressure-ventilation, in which the fan acts as a pressure-fan, and the movement of coal is unobstructed at the surface by doors or equivalent devices in drifts or shafts, the traveling and haulage thus being in the return-current.

Plain exhaust-ventilation, in which the fan acts as an exhaust-fan, the movement of coal being likewise unobstructed at the surface, and the traveling and haulage being in the intake.

Obstructed pressure-ventilation, in which the movement of coal is obstructed at the surface, in order that the traveling and haulage may be in the intake.

Obstructed exhaust-ventilation, in which the movement of coal is obstructed at the surface, in order that the traveling and haulage may be in the return.

The apparent advantages in running a fan "pressure" during working-hours, and "exhaust" between shifts, seem to justify the recognition of this combination as a standard form of ventilation. Very likely this is not a new idea, yet I do not recall any instance of it. It may be called alternating pressure-ventilation.

There may be said to be still another form of ventilation, the "continuous-current" system, which may be either "pressure" or "exhaust." It is not, however, now considered admissible in up-to-date mining-practice.

In this paper I propose to give a statement of arguments for and against each form of ventilation, and briefly to consider their weight and bearing. For such a discussion, the special conditions of each particular problem—such as whether the mines are highly gaseous, or dusty but free from gas, or both gaseous and dusty—should be stated. Moreover, the objections to either system should be separated into two classes: those which can be overcome by a reasonable expenditure, and those which are, under the conditions stated, insurmountable.

II. Arguments.

A. *Against Plain Pressure-Ventilation.*

1. *The traveling and haulage in the return-current, required by plain pressure-ventilation, are objectionable.*

In districts where coal is mined on the pitch; where the air-ways are often somewhat restricted in cross-section; where the amount of gas is so large that even considerable volumes of air are relatively small; and especially where sudden outbursts of immense volumes of gas are not uncommon, the necessity for removing the gas in the shortest time, and by the shortest route, to a point beyond further material danger, seems to make clearly impracticable that system of ventilation requiring traveling and haulage in the return.

In pitching seams, where favorable conditions or heavy expenditures on larger intakes and returns make ample volumes of air available, the objection to traveling and haulage in the return should not be so serious, although it would still have some weight on account of the possibility of outbursts of gas of unusual volume.

It is at least a plausible view that sudden outbursts of gas are largely due to cavities resulting from faults, dislocations, or disturbances of the strata, and that, relatively, the sudden outbursts of gas should be small, in drift-mining in horizontal strata, known to be free from faults or other disturbances, even though the regular evolution or transpiration of gas is large. In such mines the number of intakes and returns can usually be increased at slight expense, thus insuring a volume of air far in excess of that possible under the conditions first stated; and it does not appear that traveling and haulage in the return would be objectionable where the entire combined dust-, gas-, and smoke-content of the return is far below the lower explosive limit.

The fact that steam-locomotive haulage has been used for years, and is still used in many places, seems to show that this form of power-haulage (necessarily performed in the return-current) may be regarded as fairly safe, under suitable conditions. It is possible that the temperature of the arc resulting from electric haulage may be higher than any temperature that ever occurs in the fire-box, even with forced draft; but the occasional arc is a trifle compared with the constant passage of considerable volumes of the return-air through the necessarily high temperatures of the fire-box of the locomotive. This often causes large volumes of flame to impinge against the roof, heavily laden with the accumulations of soot, if not dust, resulting from years of haulage. The degree in which the electric arc is less seriously dangerous may represent a larger scope for electric haulage in the return-current, as compared with the steam-locomotive.

The danger of travel and haulage in the return, while clearly prohibitory in some districts, varies between wide limits, depending upon the conditions; and the value of argument No. 1 should be rated accordingly.

2. *In winter there is serious risk, with plain pressure-ventilation,*

of the air-course freezing-up at a critical moment and possibly causing an explosion.

This objection is more serious in shafts than in drifts. It is quite possible in some localities that the air-shaft may freeze tight in half a day, while a much greater length of time would ordinarily be required in drifts. But this greater margin of time is not altogether an advantage. It may easily be off-set by the carelessness which it is likely to breed.

Unquestionably the possibility of the freezing-up of the air-way is an important matter. Such an occurrence would be more serious than a breakdown. The gradual failing of the ventilation would not be as noticeable on the inside, and would therefore permit heavy accumulations of gas before the trouble was located; while on the outside, with the fan running in good order, particularly if electrically driven, at practically constant speed, and in the absence of close attention, very likely the trouble would not be suspected until word was sent from the inside, or until it was too late for remedy. A sudden breakdown of the fan, on the contrary, would be noticed and reported to the inside.

That this weighty objection, however, is insurmountable, and cannot be removed in most instances by a reasonable expenditure, does not clearly follow. In winter, the intake-air, entering the drift or air-shaft, gradually becomes warmer as it gets further underground, taking up moisture, instead of giving it up; hence, the freezing-up must be due to "drippers" from the roof, or seepage through the sides of the air-shaft, as the case may be. In drifts, the boxing-in of the air-way, so as to cut off all drippers, should be but a trifling expense; even a brick arch, which may be of lighter construction than in the haulage-drift, should be looked upon as a small matter in instances in which plain pressure-ventilation seemed otherwise advisable. It is neither so simple nor so inexpensive to stop the seepage through the sides of an air-shaft; yet the few cases in which the cost might be prohibitory should not warrant a general objection to pressure-ventilation on this score.

Where the amount of moisture is not too large, and where the fan is run night and day, reversing the fan at night so as to make it exhaust should result in thawing out at least part of the ice accumulating during the daytime in the air-way;

and the accumulation of ice thus induced during the night, in the drift-mouth or the hoisting-shaft, would then be merely a question of small expense and not of danger. Allowing 4 hr. "pressure"-run for the inspection of the fire-boss will still leave the air-course 10 hr. to thaw out, with 10-hr. shifts. With 8-hr. shifts, the air-course will have 12 hr. to thaw out.

Frequent reversing of the action of the fan would be advisable, particularly with wooden housings, if only to insure that the different doors were always in good working order in either position.

3. *Pressure-fans are objectionable on the score of possible breakdowns.*

Unquestionably the tendency of pressure-fans is to force back the gases, not only into the solid coal but also into the goaves. In case of the breakdown of a pressure-fan, the gases would suddenly be set free to the extent to which they had been previously held back by the fan-pressure, and would pass into the circulation of air, possibly with disastrous consequences. On the other hand, the tendency of exhaust-fans is to draw the gases, not only out of the solid, but also out of the goaves; and in case of the breakdown of an exhaust-fan, the gases should cease to flow out of the solid coal and the goaves until the effect of the fan-depression had been overcome by the regular evolution or transpiration of gases, thus giving every one good time to get out.

As far as these two tendencies are concerned, a point against the pressure-fan, and in favor of the exhaust, is clearly made; but the weight of this consideration is not, in the light of our present knowledge, so clear.

In some instances, at least, duplicate fans have been installed, so that there need be no serious interruption of the ventilation in case of the breakdown of one. This is unquestionably a good arrangement, and would obviate the above objection to the pressure-fan. Duplicates, however, if not really required, represent a certain amount of idle capital, and thus an unnecessary burden.

The gases upon which the fan acts are those in the advance-work or "in the solid," and those in the goaves. For a fan capable of giving a steady working-pressure of as much as 6 in.

water-gauge, there will still be a considerable drop in pressure between the fan and any given set of working-faces. If we assume this drop to be but 2 in., the net water-gauge at these working-faces will be 4 inches.

If the fan be run "pressure" in the daytime and "exhaust" at night, the total net pressure available in the morning, on reversing the action of the fan, and tending to keep the gases "in the solid," and to restrict the transpiration of gas, will be but 8 in. water-gauge, or, say, one-fiftieth part of one atmosphere. As a general proposition, this seems to be a negligible quantity, if the gases in the coal and in the surrounding strata (where mining has reached a considerable distance from the surface, with good cover) are generally under such high pressures as have been reported from actual measurement—reaching, in some instances, 32 atmospheres.

One further remark is called for. It is at least possible that a fan-pressure as high as that above mentioned may have, under some circumstances, a greater influence on the transpiration of gases than it now seems reasonable to expect. It has been reported that, in certain Austrian experiments, a decrease of pressure amounting to only ⅛ in. of mercury (or 1.7 in. of the water-gauge) resulted in an increase of the flow of gas in the proportion of 100 to 235.

The gas in the goaves may, properly enough, be supposed to be under the ordinary atmospheric pressure; the actual pressure, positive or negative, in any given section of the goaves would then depend on the fan-pressure, the system of ventilation, and the relation of the section under discussion to the other mine-workings. For the sake of simplicity, we will assume, as before, that the net effective fan-pressure at the section under discussion is 4 in. water-gauge, making the difference between the atmospheric pressure and the fan-pressure, say, one hundredth part of an atmosphere.

We will suppose an isolated section of the goaves to be 5,000 by 5,000 ft., the coal being 8 ft. thick, and that one-half only has been removed, without any clear break to the surface, and without any settling of the strata above. The supposed open space would then be 100,000,000 cu. ft.; one hundredth part of this would be 1,000,000 cu. ft. of gas that would be turned loose in the mines on the breaking-down of the pressure-fan

under the supposed conditions. Even if the gases in this section contained only 20 per cent. of marsh gas, the effect would certainly be noticeable. But even with a 6-in. water-gauge at the fan, with plain pressure-ventilation, the net fan-pressure at the section under consideration would not ordinarily be more than 2 in. instead of 4, which would make 500,000 cu. ft. of gas instead of 1,000,000 let loose by the breakdown.

These figures, while no doubt extreme, seem to show clearly that a breakdown of the fan under plain pressure-ventilation might be a most serious matter; but they furnish incidentally a strong argument in favor of clean pillar-work, with long barriers around all goaves, and as few breaks in the barriers as are absolutely required for travel and ventilating. With well-planned barriers, it should be possible to determine the effect of reversing the fan-action. All openings but one in each section of the goaves being closed, and that one reduced to a small cross-section, the effect of varying the fan-speed should be determined with reasonable accuracy. With suitably-arranged ventilation, it might be possible to throw the entire force of the fan through one opening into any given section of the goaves, when readings at the other opening (two being required in this case) would determine whether any circulation could be got through it. The effect of a breakdown of the fan might be similarly ascertained; and the matter is certainly worth the trouble of such an experimental test, in view of the great importance of definite knowledge on the subject.

On the whole, the danger of the breakdown of a pressure-fan would be a serious matter, were it not that by reversing the action of the fan at night, after the day's run is over, this question may be put in a somewhat different light. Every man being out of the mine, it would make no difference what quantity of gas was turned loose; during the night's run of the fan all the advantages of exhaust-ventilation in drawing the gases out of the goaves would be secured; in the morning, on reversing the action of the fan, fresh air would be forced into the goaves, and, so far as their arrangement permitted, would occupy, by reason of its greater weight, a horizontal stratum below the gas. While, no doubt, a certain amount of gas would be diffused into this stratum of fresh air, the daily reversal of the action of the fan should keep the goaves practi-

cally clear of large bodies of gas which might be let loose by a breakdown.

Reversing the fan-action would not necessarily interfere with a night-shift, if sufficient time could be secured between the shifts, as, I think, could be readily done in most cases.

One of the strong points in favor of plain pressure-ventilation is that the reversal of the fan-action at night secures, in addition, many of the advantages of plain exhaust-ventilation. On the other hand, such a reversal is out of the question if plain exhaust-ventilation is employed in the daytime. If the fan is run as an exhaust by day it must be so run at night also.

4. *Pressure-fans are more liable to be wrecked in case of explosions.*

It is so simple a matter to place the fan well out of line of the air-course that this argument is without serious weight.

B. *In Favor of Plain Pressure-Ventilation.*

5. *Positive or pressure-fans hold "gas-blowers" in check better than exhaust-fans.*

As shown under Argument 3, this proposition lacks substantial grounds. It may, however, receive a small value in the discussion, partly out of deference to what seems to be the general opinion.

6. *Pressure-fans have the advantage in times of falling barometer.*

A detailed consideration of this interesting argument would occupy more space than is available here. I content myself, therefore, with the following hints:

If the alleged dangerous increase in the volume of gas to be handled underground, which often seems to accompany a falling barometer, be due to some force or influence as yet unknown, and therefore not subject to control, why would it not be the safest course to stay out of the mine until the barometer gets through falling?

But if this alleged increase in gas is merely that which may result from the decrease of atmospheric pressure, then, it seems to me, the practice of "alternating" pressure-ventilation would result every morning, on reversing the action of the fan, in an artificial rise of the inside barometer, about sufficient to offset any ordinary fall of the outside barometer, which would be likely to have occurred the day before.

The fact that a pressure-fan can be increased in speed to

meet such an emergency will lose its force as a basis for this argument if steam-engines are replaced by constant-speed electric motors.

7. *Positive or pressure-fans keep fires in check better than exhaust-fans.*

This should depend somewhat upon the location of the fire, with regard to the mine-workings.

8. *Under plain pressure-ventilation, when the air is properly split, an explosion should be less disastrous.*

While there is some evidence which seems to favor this proposition, it will be, very likely, disputed by many.

9. *In shallow mines, pressure-fans may force considerable volumes of gas to the surface through pillar-breaks.*

This seems to be of minor importance even in the instances to which it applies. The danger, at a critical moment, of losing large volumes of air by forcing them to the surface instead of through the mine might easily offset any supposed advantage on the score of gas. I see no reason for giving any weight to this argument.

10. *Pressure-fans require less power.*

While this proposition may be demonstrable on paper, it is not worthy of serious consideration under the conditions, and for the purposes, of actual practice.

11. *Pressure-fans are less fouled, because traversed by fresh air only.*

This argument scarcely deserves to be put on the list. It is not pleasant to climb around inside an old exhaust-fan; but this occasional annoyance to an individual has no influence upon the cost or the efficiency of mine-ventilation.

C. *Against Plain Exhaust-Ventilation.*

12. *Exhaust-fans necessitate, in winter, the daily removal of ice from the drift-mouth, or the hoisting-shaft, as the case may be.*

This argument also is of minor importance, and should have no appreciable bearing on the selection of the right system. It involves, not elements of danger, but only a trifling increase in daily operating-expenses.

13. *Exhaust-fans, in winter, make the temperature uncomfortable at the head of the hoisting-shaft.*

This objection will hardly be appreciated by those who have not experienced its force on an extremely cold day; and unquestionably it is trivial, as affecting the selection of a system of ventilation indicated on more important grounds to be, on the whole, the best.

14. *In shallow mines, exhaust-fans may draw into the live workings considerable volumes of gas which would otherwise find their way, through pillar-breaks, to the surface.*

This objection is not, as might appear at first sight, the reverse of No. 9. As to both gas- and air-leakage, it bears against exhaust- as compared with pressure-ventilation.

15. *Exhaust-fans not only keep much the larger portion of the haulage-dust in the mine, but distribute it throughout the workings.*

Did space admit, the adequate discussion of this proposition would include: the classification of dust with respect to origin and position; the effect on the dust-problem of plain pressure-ventilation, plain exhaust-ventilation, and continuous-current ventilation; methods for the prevention, removal, and laying of dust; the minimum amount of dust required to produce explosive effects; a tentative analysis of a gas-explosion and the determination of its essential features; a classification of dust with respect to its behavior in explosions; a tentative correlation of the dust-explosion with the gas-explosion; and a consideration of smoke as one form of dust. Omitting these details here, I may say that they seem to me to warrant the following deductions:

That the tendency of plain pressure-ventilation is to carry out of the mine the larger portion of the finer grades of haulage-dust, while that of plain exhaust-ventilation is not only to retain the larger portion of such dust within the mine, but to distribute it throughout the workings; and that gas is not essential to dust-explosions. It should be neither difficult nor expensive to determine the extent to which these deductions are correct; and, in the absence of such determination, the danger from dust should not be lightly balanced against minor considerations.

D. *In Favor of Plain Exhaust-Ventilation.*

16. *Exhaust-fans have the advantage in times of high barometer.*

I see no reasons for giving to this assertion any value whatever.

E. *Against Obstructed Pressure-Ventilation.*

17. *With obstructed pressure-ventilation, the ventilating-doors at the drift-mouth, or their equivalent in shafts, may be expensive as well as inconvenient, and may fail at a critical moment.*

While the obstruction to the movement of coal at the surface, necessary with pressure-ventilation, in order to avoid traveling and haulage in the return, should receive due weight as a cause of possible increase of danger in the case of drifts, or of both expense and inconvenience in the case of shafts, this objection seems, on the whole, to be of minor importance. On the other hand, the advantage of this system over plain exhaust-ventilation is not marked.

F. *Against Obstructed Exhaust-Ventilation.*

18. *Obstructed exhaust-ventilation involves the same expense, inconvenience, and risk of failure as obstructed pressure-ventilation.*

Under this head, also, limited space forbids consideration in detail. Plain pressure-ventilation with electric haulage seems to be better than obstructed exhaust-ventilation; but with steam-locomotive haulage, in some instances, the advantage would be clearly in favor of the latter.

III. General Conclusions and Comparisons.

It is obvious that the application and force of each of the foregoing arguments must vary with the circumstances of each case, and also that the bearing of an argument may sometimes be in favor of one system, but not to the same extent unfavorable to another. For instance, the weight assigned to No. 1 in a comparative statement would be equally against "plain pressure" and in favor of "plain exhaust" ventilation; and the same would be true of No. 2 and No. 3, but for the consideration that a third system, that of alternating pressure-ventilation, may be better than either. These illustrations may suffice to indicate the appropriate method of comparative estimation.

Many of the propositions stated above still lack the experimental quantitative determination which, in my opinion, ought to be made and could be made with reasonable ease and approximate accuracy. In the absence of such determinations, the weight of such propositions must be a matter of personal

opinion. Nevertheless, a comparison of the opinions of competent practicing engineers would be interesting and valuable; but it seems to me that it would be most advantageously expressed in quantitative form; that is to say, in numbers rather than adjectives—with such explanations, of course, as each contributor may find necessary. For such a quantitative comparison, some one principal argument in favor of each system of ventilation may be assumed as a basis and rated at, say, 100; the other arguments, *pro* and *con*, receiving such numerical values as they are deemed to deserve.

The following tables are given, subject to criticism, as samples of this method applied to the arguments above stated, and representing my personal estimation of the bearing and force of each:

TABLE I.—*Plain Pressure vs. Plain Exhaust.*

Argument No.	Plain Pressure.		Plain Exhaust.		Remarks.
	For.	Against.	For.	Against.	
1	....	50	50	....	Probably a maximum.
2	....	10	10	....	May be reduced to nothing.
3	....	5	5		
4	....	....	....	....	No practical value.
5	3			3	
6	10			10	
7	5			5	
8	10			10	Doubtful value.
9	....	....	....	....	Not in the case.
10	....	....	....	....	No value.
11	....	....	....	....	No value.
12	1			1	
13	1			1	
14	....	....	....	....	Not in the case.
15	100			100	Basis of comparison.
16	....	....	....	....	No value.
17	....	....	....	....	Not in the case.
18	....	....	....	....	Not in the case.
	130	65	65	130	

Conditions: Drift-mines; the overlying strata being largely hard, tough sandstones, between fairly firm shales, and free from faults or breaks of any kind, so that the evolution of gas, while considerable, is fairly regular, and the chance of heavy outbursts is small; the movement of coal unobstructed at the surface; electric haulage; air-ways driven at least double, thus making available large volumes of air; capacity of fan ample, with relay-fans, if necessary; the air generally well split; no doors that can short-circuit large sections; soft bituminous coal containing 20 per cent. of volatile matter; the sections of mining-work separated by barrier-pillars. The question of dust (Argument No. 15) having the value of 100.

In this instance, out of a total of 195 points in either case, 130 are in favor of pressure-ventilation, and only 65 against it. The values given to most of the arguments, as compared with Nos. 1 and 15, will no doubt be considered unreasonably low.

TABLE II.—*Plain Pressure vs. Plain Exhaust.*

Argument No.	Plain Pressure.		Plain Exhaust.		Remarks.
	For.	Against.	For.	Against.	
1	....	100	100	....	Basis of comparison.
2	....	15	15	....	Should be maximum average.
3	....	5	5		
4	....	....	....	....	No value.
5	5			5	
6	10			10	
7	5			5	
8	10			10	Doubtful value.
9	....	....	....	....	Not in the case.
10	....	....	....	....	No value.
11	....	....	....	....	No value.
12	3			3	
13	2			2	
14	....	....	....	....	Not in the case.
15	5			5	
16	....	....	....	....	No value.
17	....	....	....	....	Not in the case.
18	....	....	....	....	Not in the case.
	40	120	120	40	

Conditions: Shaft-mines, the mining being on the pitch; the strata badly broken and faulted; heavy outbursts of gas not only of common occurrence, but always probable; gas generated in large quantities by the advance-work; the ventilation, while reasonably safe, requiring large volumes of air; the air-ways single, and of only fair dimensions; the coal, anthracite; the cover necessarily varying, but, say, from 300 to 700 ft. thick; and Argument No. 1 (concerning traveling and haulage in the return-current) receiving the value of 100.

Out of 160 points, 120 are against plain pressure-ventilation, with only 40 in favor of it; or, 3 to 1 against plain pressure-ventilation, or in favor of plain exhaust, under the conditions specified.

In Table III., the balance, although slight, is in favor of obstructed pressure-ventilation; arguments No. 2 and No. 7 should perhaps have been omitted.

The balance seems to be against obstructed exhaust-ventilation as a general proposition, as shown in Table IV. With steam-locomotive haulage, for instance, the position of the intake, on mines opened from the outcrop, may make plain pres-

TABLE III.—*Obstructed Pressure vs. Plain Exhaust.*

Conditions: The same as in Table II.

Argument No.	Obstructed Pressure.		Plain Exhaust.		Remarks.
	For.	Against.	For.	Against.	
1	100		100	....	Basis of comparison.
2	15		15	....	Should be maximum average.
3	....	5	5	....	
4	....	....	....	....	No value.
5	5			5	
6	10			10	
7	5			5	
8	....	10		10	Doubtful value.
9	....	....	....	....	Not in the case.
10	....	....	....	....	No value.
11	....	....	....	....	No value.
12	....	3		3	
13	....	2		2	
14	....	....	....	....	Not in the case.
15	....	5		5	
16	....	....	....	....	No value.
17	....	10	10		
18	....	....	....	....	Not in the case.
	135	35	130	40	

TABLE IV.—*Plain Pressure vs. Obstructed Exhaust.*

Conditions: The same as in Table I.

Argument No.	Plain Pressure.		Obstructed Exhaust.		Remarks.
	For.	Against.	For.	Against.	
1	....	50		50	Probably a maximum figure.
2	....	10		10	May be reduced to nothing.
3	....	5	5		
4	....	....	....	....	No value.
5	3			3	
6	10			10	
7	5			5	
8	10		10	....	Doubtful value.
9	....	....	....	....	Not in the case.
10	....	....	....	....	No value.
11	....	....	....	....	No value.
12	....	....	....	....	Not in the case.
13	....	....	....	....	Not in the case.
14	....	....	....	....	Not in the case.
15	100		100	....	Basis of comparison.
16	....	....	....	....	No value.
17	....	....	....	....	Not in the case.
18	2			2	
	130	65	115	80	

sure ventilation clearly inadvisable, if not altogether impracticable. This, however, would require for exhibition a special table.

Such tabular comparisons may be varied in many ways; and special problems may be indefinitely multiplied. A special table might be required for steam-locomotive haulage, the conditions of which give special value to considerations otherwise comparatively unimportant. But that system of haulage can hardly be regarded as possessing a live interest at present, since it is rapidly giving way to superior modern methods.

It seems to me, however, that if we could make a beginning by settling relations and values, in the light of our present knowledge, for a few leading and typical sets of conditions, the numberless variations presented by practice might be similarly treated without much difficulty by modifying appropriately the numerical values attached to the factors of the problem.

The American Institute of Mining Engineers and the Conservation of Natural Resources.

BY JOHN BIRKINBINE, PHILADELPHIA, PA.

(New Haven Meeting, February, 1909.)

AWAKENED public interest in efforts to conserve natural resources will certainly be appreciated by the members of the American Institute of Mining Engineers, and a discussion upon conservation may well form a part of one of the Institute meetings, for the records of papers and discussions which appear in the 38 volumes of its *Transactions* are replete with evidence of efforts in this direction.

Every process which reduces the fuel consumed or the power applied per unit of product; every utilization of waste material, or employment of that formerly rejected, to obtain something of value; every feature of construction or operation which lessens the labor required or increases the output per employee, is an advance in the direction of true conservation. This may not appear from the records of late conferences or from published articles, in which the effort seems to be concen-

trated upon reducing the consumption, and presenting to the people of the country estimates of exhaustion of natural resources.

I rejoice in the awakening of public interest in a matter so important for the present and future welfare of our nation: I appreciate the possibility of much good following conferences of the Governors and delegates from various States and from national societies; and I recognize the interdependence, not only of different States, but of contiguous nations. In fact, the marvelous development of transportation by land and by water, and the decreasing rates at which commodities can be moved great distances, have in many cases practically obliterated geographical boundaries, and nearly every country on the globe can either market its product in other lands or draw upon them for raw or manufactured materials; hence, conservation is an international subject.

In the report of the proceedings of the national conferences the exhaustion of our natural resources has been the keynote; and while we should frankly consider this, studying the present and future, we should not forget that the utilization of these natural resources means progress, and that the uplifting of the nation has been due to their abundance, and their application to produce materials necessary for the development and advancement of the country.

That much waste has attended the growth of our industries, and that great waste is now too prevalent, none will gainsay. Forests, which in the early settlement of the country obstructed the cultivation of the soil, were looked upon as undesirable and reduced as speedily as possible; but in late years willful destruction of forests has been far too prevalent. In Texas I have seen beautiful and choice specimens of hard-wood timber, such as walnut and oak, girdled to advance their death and destruction, and thus make way for the plow and harrow.

A visit to inspect a coal-outcrop in the State of Washington in 1889 was made abortive by fire, started among the monarchs of the forest, and consuming many superb trees, to clear for the site of a town which is not yet on the map. At the present time visitors to Florida will notice immature yellow-pine trees "boxed" to such an extent as to leave but a few strips of bark connecting the upper portion of the tree with the roots, pre-

venting them from achieving full maturity; and in the same State, under the protection of law, thousands of good trees fall a prey to flames kindled to make better grazing for cattle, whose total money-value is much below that of the trees destroyed.

In but few portions of the country has public sentiment been educated sufficiently to bring about the punishment of those who purposely or carelessly set fire to woodland, and in many States tax-assessments, increasing with the age of the timber, encourage owners of wooded areas to cut the trees as soon as the timber can be marketed.

There have been, and still are, enormous losses in cutting timber for lumber purposes, and in fabricating the lumber, attested by the mass of tops left in the clearings to intensify any future fire, by the flames from the "hells" at saw-mills, or by great piles of débris obstructing water-ways. But, recognizing all these evidences of waste, we must admit that the lumber industry has added greatly to the wealth of the country; that many economic methods have been applied; that substitutes for wood or applications of refuse have been introduced; and that cheap lumber has aided greatly in the settlement of our land.

The old blast-furnace plants operated with charcoal were classed among the destroyers of the forests, but at present some of the tracts of woodland, which were maintained as such to keep up a supply of charcoal, represent large contiguous areas which have been acquired by States as forest reserves.

Forestry has been a feature in discussions before the Institute, and such utilizations of waste as the application of saw-dust and mill-refuse for the production of gas for metallurgical purposes in Sweden are chronicled in the *Transactions*.

When the Institute was in its infancy, the country was supplying 51,500,000 tons of coal annually, of which nearly one-half was anthracite, and the pig-iron production was but 2,500,000 tons. In 1907 we supplied 480,000,000 tons of coal, of which anthracite represented less than 20 per cent., while the pig-iron output reached approximately 26,000,000 tons.

In the interval, improved methods, which have cheapened the production and greatly increased the output of coal, of iron-ore, and also of all other minerals, have been made possible processes, many of which first secured public attention

through the Institute's *Transactions.* This also is true as to the use of coals which formerly were considered undesirable, gas-producers, by-product coking, gas-engines, etc., each of which has received recognition in our papers and discussions.

As features of conservation, the reduction of mine-waste and the recovery of coal from refuse have been well considered, for in 1907 the quantity of anthracite coal used, which was of sizes unmarketable in 1872, or which was recovered from waste-piles, approximated the entire production of this mineral in 1872.

A comparison of the conditions is well shown by the growth of the steel industry, which, when the Institute was organized, represented a production of 143,000 tons, of which 70 per cent. was obtained by the Bessemer and less than 3,000 tons by the open-hearth process. The output of Bessemer steel reached 1,000,000 tons in 1880, in which year the open-hearth product first exceeded 100,000 tons. In 1900 the steel-output of the country was more than 10,000,000 tons, divided practically into two-thirds Bessemer and one-third open-hearth; but in 1907 more than 23,000,000 tons of steel was produced, the proportions of Bessemer and open-hearth being nearly equal.

Much of this marvelous growth has been due to improvements described or modifications suggested by the Institute discussions, which are to be credited with a large part of the increase in product. But, apart from the quantities produced, we have to consider the economies in fuel-consumption, the reduction of waste, the substitution of mechanical appliances for manual labor, and other features which have gone far towards making the rapid growth possible.

A retrospect of blast-furnace practice shows wonderful development in producing-capacity, accompanied with equally remarkable fuel-reduction per ton. Although the figures demonstrate a ten-fold growth of pig-iron manufactured, it is doubtful if the amount of fuel consumed in making pig-iron in 1907 was eight times that consumed in smelting the smaller output of 1872. Equal or greater economies can be observed in advanced lines of manufacture, demonstrating that our progress in iron and steel, while necessitating enormous drains upon our fuel-supplies, has aided in their conservation by reduced fuel-consumption per ton of manufactured product.

If the increased utilization of other minerals were similarly discussed, corresponding advances, not only in quantity, but also in economical conditions, would be shown.

The fuel, the minerals, and the manufactured and agricultural products of the country have been moved by railroad-transportation, which, during the life of the Institute, has been multiplied four times in mileage, while each mile of railroad is to-day doing many times the business that it did 36 years ago, but doing it at a lower cost and with smaller fuel-consumption per ton-mile. Similar advances have taken place in water-transportation.

The composition and physical treatment of steel for rails, etc., and of iron for car-wheels, etc., occupy many pages of the Institute's *Transactions*, and may be taken as indicative of the manner in which our members have aided in the promotion of true conservation.

The commercial uses of electricity, for power, light, and transportation, were practically unknown at the birth of the Institute, this marvelous development having taken place within three decades; while electro-chemistry and electro-metallurgy on commercial scales are also innovations which incidentally have brought about water-power improvements by hydro-electric plants of great power. The creation of a Portland-cement industry, with a yearly output amounting to 50,000,000 barrels, may be mentioned among the newer features of industrial progress.

Waste in petroleum characterized the early history of each producing district, but in the utilization of this material and the recovery of its by-products there have been wonderful advances.

To the foregoing suggestive instances of the progress already made, each member may add from his own experience. My purpose in mentioning them is to demonstrate the claim that proper utilization is true conservation. I have not attempted a complete review of the 38 volumes of our *Transactions*, covering all the particulars of the aid we have given to such utilization. Nor do I desire in the least to minimize the importance of estimating the extent of our national reserves and the possibility of the exhaustion of any of them. But eloquent prophetic descriptions of a treeless country, with scant water-

supply, little if any coal or iron-ore, and eroded, barren farm-lands, as well as extravagant statements of wholesale waste, should be considered in connection with the progress already made and the economies already introduced.

President Roosevelt's assertion that the increased consumption of coal by the United States in the year 1907 over that of 1906 equaled the entire production in 1876 is statistically supported; but he might have added that, in the absence of economies introduced in the 31 years' interval, this increase would have been more than doubled.

The assertion that the iron-ore supply may last but a short time, unless leaner ores are used, is not based on a *résumé* of estimated reserves, except in certain sections; but, if leaner ores must be used, the United States will be at no disadvantage with other iron-producing countries. Not one of the larger contributors to the world's supply of pig-iron feeds to its blast-furnaces ore-mixtures as rich in iron as are those smelted in the United States.

Anticipating that the sudden awakening of popular interest in conservation may be short-lived, unless an appreciation of utilization is associated with it, I hope that this interesting and important problem will be treated, not as a new cult, but as a practical development for which able men have labored conscientiously, persistently, and not unsuccessfully, for many years. The members of the Institute are especially bound to claim for many illustrious men among its members who have passed away, as well as many who are now living, the credit due for devoted, disinterested, and most effective, though not theatrical and sensational, work, which accomplished more in real results of national economy than any vague, indiscriminate, and undirected popular enthusiasm, or any crude and hasty legislation, however patriotic in spirit and purpose, could reasonably be expected to effect.

It would be invidious to select from our past and present membership the names to which such honor is due. But no one will withhold his assent and applause from my mention of one shining example, Eckley B. Coxe, a most accomplished mining engineer of his generation, one of the founders of the Institute; for 25 years (until his death) a member; for ten years a Vice-President, and for two years a President.

At the meeting in Wilkes-Barre in April, 1871, at which the Institute was organized, Mr. Coxe was made the Chairman of a Committee on the Waste of Anthracite Coal. The results of his work will be found largely in the reports of the Pennsylvania State Commission, which he caused to be appointed, and the expenses of which he paid. At that period, the Institute was not pecuniarily able to make such publications. But, according to Mr. Coxe's repeated declarations, and within the knowledge of all his colleagues who were acquainted with the circumstances, his great labors, and their great results, were based upon the duty originally laid upon him by the new-born American Institute of Mining Engineers. Without going into further details here, it is not too much to say that what Mr. Coxe accomplished, by unstinted labor and the expenditure of many thousand dollars of his private fortune, in that field of true and immediate "conservation," was more important than any subsequent achievement in the same direction. To this day, the improvements introduced by Eckley B. Coxe, and adopted by the great corporations engaged in the mining and transportation of anthracite, constitute the chief, if not, indeed, the only real measures for the conservation of our anthracite to which the patriotic political economist can "point with pride." And they are not likely to be surpassed by any experiments of noble but less practical enthusiasm.

In short, the work of this Institute has been, as it should continue to be, the patient and intelligent dealing with facts, and the practical prosecution of wise methods of improvement, without appeals to the heat of uneducated sentiment or temporary gusts of popular demand. These agencies are valuable as providing the much-needed power (whether of steam or air); but it is still the duty of the engineer wisely and calmly to run the machine or to sail the ship.

Conservation of Natural Resources.

BY JAMES DOUGLAS, NEW YORK, N. Y.

(New Haven Meeting, February, 1909.)

IN discussing the waste upon which hinges, or is supposed to hinge, so largely the preservation of our national resources, the conclusions reached would be more reliable if actual experience were consulted, and fewer deductions were drawn from general statements, which are often the product of the imagination.

It cannot be questioned that the value of by-products has not been sufficiently appreciated by us, and that our tardiness in recovering the useful ingredients of the escaping gas of our coke-ovens is one of the most glaring instances of shortcoming in that direction. And yet even for that sin there is some palliation in the immature condition of affiliated industries. I presume that it is admitted, without argument, that, except under very exceptional conditions, all the elements cannot be recovered from most of the ores or natural products which we treat. While it is a shame that the by-products from our coke-ovens should be dissipated, Edward W. Parker's report to the U. S. Geological Survey for 1906 [1] supplies a fairly good excuse in justification of this appalling waste. He says (pp. 773 to 774.):

"What has been already commented on in previous reports about the slowness of manufacturers to change from the better known but wasteful beehive practice to the by-product recovery method of coke manufacture is particularly emphasized in the statistics presented in this chapter. For it would appear from the table following that the construction of by-product ovens had about come to a standstill, especially when the records for the preceding five years are taken into consideration. At the close of 1901, when there were only 1,165 by-product ovens completed in the United States, there were 1,533 in course of construction, 498 of which were completed during the following year. At the close of 1902, 1,346 retort ovens were building, 293 of which were added to the completed plants in 1903. At the close of 1903, 1,335 new ovens were building and 954 of these

---

[1] *Mineral Resources of the United States for* 1906, U. S. Geological Survey (Washington, 1907).

were put into blast before January 1, 1905, at which time 832 new ovens were in course of construction. At the close of 1905 there were only 417 new ovens building, and at the close of 1906 new work was limited to 112 Otto-Hoffmann ovens, which were being added to the 260 ovens already built at Johnstown, Pa., by the Cambria Steel Company. These new ovens were completed and put in blast in February, 1907.

"This condition is somewhat difficult to understand when the economies effected by the use of retort ovens have been so clearly demonstrated. These economies consist not only in the higher yield of coal in coke, but in the recovery of the valuable by-products of gas, tar, and ammonia. One of the reasons that has been assigned for the comparatively retrogressive condition exhibited by the statistics for 1905 and 1906 (comparison being made with beehive oven construction, 5,893 new beehive ovens having been completed in 1906, with 4,407 building at the close of the year) is the lack of a profitable market for coal tar, and yet the United States is importing coal-tar products to the value of several million dollars annually, while the development of the fuel-briquetting industry has been held back because of the lack of assurance of a steady supply of coal-tar pitch for a binder, and users of creosoting oils for the preservation of timber complain of an insufficient domestic supply of this product of coal-tar distillation."

The truth is that one branch of industry is so dependent upon another that there must be equal progress along the whole line of industrial life, if complete recovery of all the available elements of our natural resources is to be effected. The chemical industry must keep pace with the mining and metallurgical industry. We may be moving too slowly in that direction, but we can distinguish a steady movement towards this needful co-operation. It is encouraging, for instance, to find that the waste gases from the furnaces of the Tennessee Copper Co. are being turned into sulphuric acid for the manufacture from Southern phosphates of the super-phosphates which the fertilizers of the Southern cotton-fields need. Failing this mutual relation between the metallurgist of Tennessee and the chemical manufacturer, the blame should not rest entirely upon the metallurgist for wasting that for which, heretofore, he has been unable to find a market. The same justification exists abroad as in this country for similar waste in other branches of industrial activity.

It is nevertheless true that legal compulsion alone has driven manufacturers to introduce improvements and economies which were demanded by public safety, and which have redounded to the benefit of the reluctant corporations. In Germany and England the disposal of noxious vapors and noxious liquors has been required of the manufacturers, but their compulsory

removal from the atmosphere and the water has resulted in their conversion into useful products, and the building-up of new technical industries. An agitation is springing up in the West against the fumes from smelting-works being turned loose into the atmosphere. While in some cases the injury done to vegetation may have been falsely attributed to the smoke from metallurgical works, the agitation has been followed by some good results. For instance, the Mountain Copper Co., having been driven out of Shasta county, Cal., by the farmers, has erected chemical-works as an annex to its smelter at Martinez, on San Francisco bay. Here, as elsewhere, manufacturers are reluctant to go to the heavy expense involved in abating such nuisances, even though they may know that in the end the abatement will be profitable. As far back as 1881 Mr. Vivian admitted that in recovering 47 per cent. of all the sulphurous acid emitted from his furnaces in Swansea, he condensed 3,666 tons of oil of vitriol at a great profit. This valuable asset, though he does not so state, was secured in spite of bitter opposition on the part of those who were ultimately the most benefited by it. One looks with wonderment at the clouds of valuable fumes which float from the New Jersey shore over Staten Island to the sea, instead of flowing inland as acid to the chemical manufacturers in the neighborhood.

Our industrial development, however, has reached such a state of advancement, especially in the densely-populated portion of the country, that however averse some of us may be to expend a large share of our profits in improvements, designed primarily to relieve the public of nuisances, we must submit whether we will or not. And having obeyed the mandate of the law, not many years will elapse before we come to realize that what we do under compulsion is as much for our own good as for that of our neighbor.

I promised, however, to confine myself in my remarks to matters of experience. I have been identified with the copper-interests of the Southwest since 1881. Though the Southern Pacific railroad had only just traversed the territory, mining was immediately stimulated by railroad-transportation, and the Copper Queen Co., at Bisbee, the Old Dominion Copper Co., at Globe, and the Lezinskys (the predecessors of the Arizona Copper Co.), as well as the Detroit Copper Co., were actively

at work at Clifton. All three of the most productive districts, therefore, of southern Arizona were being explored, and, through the influence of the railroad, vigorously exploited, at that time. But none of them were situated on the main line, or were linked to the trans-continental road by branches. The Copper Queen was 60 miles from its nearest railroad-station, Benson; the Old Dominion was 140 miles from either Wilcox or Bowie; and the mines of the Arizona Copper Co., and the Detroit Copper Co., were 80 miles from Lordsburg. Coke and supplies had to be hauled in and copper teamed out those long distances.

The ores in all three camps were thoroughly oxidized. At the time this was supposed to be a condition of the highest advantage, upon which the only possibility of economical treatment depended; and not without good grounds, for the tedious methods of treating sulphide ore, so expensive in labor and fuel, were still practiced. We all therefore imagined in our shortsightedness that the day of doom for the copper-interests of southern Arizona would date from the transition from oxidized to sulphide ore. Of the three districts, the only prosperous one during the succeeding 15 years or so was the Warren, and for reasons which we now more clearly appreciate than we then did. The ores of the Copper Queen, or rather such of them as were then selected for treatment, were self-fluxing. They contained about 10 per cent. of copper. The slags of that period, which we are now re-smelting, contained about 2.5 per cent. of copper. Assuming the slags to represent 65 per cent. of the charge, about 16 per cent. of the total copper-content was being stored away in them. Less-favorable conditions, however, existed at both Globe and Clifton. The ores of both these districts were extremely siliceous, and the furnace-charge of ore had to be diluted with from 40 to 50 per cent. of limestone. The siliceous ores as treated were probably of about 12 per cent. The furnace-charge was reduced by fluxing to between 7 and 8 per cent. of copper. The old slags —65 per cent. of the total charge—yield at Globe about 3.5 per cent., and therefore must have carried from 30 to 32 per cent. of the total copper fed into the furnaces. We have re-treated all the old slags of the Detroit Copper Co., at Morenci, near Clifton, and know that they carried on an average 4.5 per

cent. of copper, and must therefore have contained at least 40 per cent. of the copper in the ore. At neither Clifton nor Globe was the dust collected, which probably represented a loss of another 5 per cent. Considering the high cost of fuel and labor, it is not to be wondered at that neither the Old Dominion, the Arizona Copper Co., nor the Detroit Copper Co., was financially successful for the first 15 or 16 years of their existence. It was not until all the richer carbonate ores had been wasted by being largely converted into slags that the companies recognized that their salvation depended upon securing sulphide ores; upon making metallic copper through the medium of matte, and throwing away less copper in their slags. So little, however, was this fact appreciated at first that we all envied the Arizona Copper Co., because it could turn the San Francisco river into its works and granulate and wash away this valuable refuse. And when the Old Dominion mine struck large volumes of water, the Old Dominion Co. committed the same act of folly, washing its 3.5-per cent. slags into Pinal creek.

Had the companies realized the losses they were incurring and the only remedy applicable, they would have been obliged to close both mines and furnaces; for except at the Copper Queen, where sulphide ores were encountered within three or four years after the mine was opened and were considered a nuisance, heavy sulphides are rare. Though the Old Dominion Consolidated Co. has explored its property to the 16th level, between 100 and 200 tons daily are imported from California and Bisbee, the company's own mines producing only about 60 per cent. of the sulphur required by the furnaces. And at least one of the Clifton smelting-companies is obliged to draw daily from abroad by railroad about 160 tons of sulphides high in sulphur and low in copper. It follows, therefore, that there was no alternative in the early days between either suspending operations or making copper in the wasteful manner which the companies then pursued.

Looking at the situation from the stand-point of to-day, if we place the advantages and disadvantages side by side, we have on the side of the advantages:

1. The experience which was gained during that long period of adversity, which is now being turned to good account, not only by the original companies, but by the many other enter-

prises which have entered the same field and are profiting by the losses of the pioneers.

2. The southern portion of the territory has increased in population and in wealth, mainly through the exertions of these copper companies, even while they were losing money on the copper produced. They not only employed thousands of men, but they made a market for the agricultural development of the small amount of arable land within reach of the mines. Had the mines of Globe and Clifton not been operated because pecuniarily unsuccessful, and had not the shareholders been willing to accept hopeful promises in lieu of dividends, Arizona would not to-day be making an unanswerable plea for admission to the Union as a State.

3. The ultimate success has been due to the advent of the railroad; for railroads are seldom built into unproductive regions in the expectation of creating traffic that does not exist.

If we turn to the disadvantages, they are of course palpable. At the present time, when we are matting our copper-ores instead of making black copper direct, the slags from those three groups of copper-furnaces run from 0.4 to 0.5 per cent. of copper. Even when the slags are re-treated, copper in the slags resulting from the slag-treatment runs higher than in slags from the treatment of ore, owing to the difficulty of reducing silicates. Thus, when the slags are re-treated, there is the double waste of fuel and the double waste of labor.

Even supposing that our economic system were different, and that necessity did not drive public corporations to utilize wastefully the resources they acquire, I think that the balance of advantage to the country at large, as well as to the district, would indicate that it is better to make progress and thereby gain experience, even at the expense of such waste as I above indicate, rather than stand still and do nothing, in the hope of more favorable conditions being brought about by Providence rather than by our own efforts.

Certain lessons, however, the above recital of experience teaches. One of them is, never to throw away anything that contains material of any value, even though it may seem to be valueless. The time inevitably and invariably comes when, through improved conditions or better methods, what was waste to one generation becomes of value to another. Most of

the filling of the old stopes in the Copper Queen mine and in the Old Dominion mine has already been re-treated. In the case, therefore, of sulphide ore, which is too lean to handle, it should be stored underground rather than exposed to the weather at the surface. I am not sure whether we are justified in ballasting our railroads with the slags which we are making now—lean as they are. One cannot see how 0.5 per cent. of copper and a little gold and silver can possibly be recovered to any advantage, and yet the future may reveal secrets which will convert such impossibilities into possibilities. The slags from the iron blast-furnaces, which were deemed valueless a generation ago, are made into hydraulic cement to-day.

We all recognize the waste that has resulted in the past from washing away gold-tailings, which often ran several dollars in gold to the ton. Had they been impounded, the minerals now, through weathering, would be in the fittest possible condition for cyaniding, and would give up to this process their residual values to within a trifle of their contents. The same rule of preservation should be applied to the tailings from copper-works. The sulphides, no matter how small their percentage, slowly decay, and give off their copper as soluble sulphate, which can be precipitated on scrap-iron at a very inconsiderable cost. If the locality be such that these waste materials can be stored, care, and some outlay, if necessary, should be expended in their preservation.

There seems, however, to be a fascination in contemplating loss rather than saving, and while we cannot exaggerate the follies of waste, it is not fair to the profession to overlook the efforts that have been consistently made within the last three-quarters of a century, and are still being made, to eliminate waste. One of the anomalies, however, of the problem is that the accused mining and technical engineers compose the only section of the public which really appreciates the cost of waste and tries to save.

The recovery of heat-units in our domestic fire-places and furnaces is far less than the recovery of heat from coal burned under our best boilers, when measured as power generated in our steam-engines. And the waste in our kitchens and at our tables involves a greater national loss than the waste in our coal-mines. In the one case the people at large are making no

effort to minimize it, while every technical man of repute is putting his best endeavors into devising means of getting the highest efficiency out of nature's forces, with a view to turn nature's resources indirectly to the greatest good for the greatest number.

If we look backward to what has happened within our own day and experience, we may justly feel some resentment at the harsh criticism which is now being so generally aimed by the press and the public at technical men. And this is partly true likewise of the strictures so indiscriminately passed upon the corporations which are instrumental in developing the country's natural wealth.

In the middle of the last century, less than one-half of the iron made in this country was smelted with anthracite, and the balance with charcoal or charcoal and coke.[2] The devastation of the forests was awful. Pearse[3] gives the consumption of wood in Berks county, Pa., in making 19,000 tons of charcoal-iron in 1828, 1829, and 1830 at 250,528 cords. To secure this amount about 8,000 acres of the finest forest-land in the country must have been stripped. In England, where most of the iron was made with coke as fuel, at the same date, and until 1875, there were consumed from 35 to 37 cwt. of coke per ton of pig-iron. In 1875, when the Whitwell stove was introduced to heat the blast, the quantity of fuel consumed was reduced by 3 or 4 cwt. By improved mechanical and metallurgical appliances that consumption in the Middlesbrough district is now lowered to 22 cwt.[4]

This saving of fuel in the blast-furnace has, in this country as well as in Europe, been effected through the sleepless activity of metallurgists and engineers, by modifying the size and shape of the great iron stacks, increasing and regulating the temperature and the pressure of the blast, and by the introduction of appliances for utilizing the waste heat. The difference between the 37 cwt. of coke formerly needed to make a ton of pig-iron and the 22 cwt. now consumed, multiplied

[2] In the *Iron Manufacturer's Guide* (1866), Lesley gives the total production in 1854 at 724,833 tons, of which 417,123 tons was charcoal- or charcoal-and-coke iron.

[3] *Concise History of the Iron Manufacture*, p. 156.

[4] A Description of Messrs. Bell Brothers' Blast-Furnaces from 1844 to 1908, and other papers, *Journal of the Iron and Steel Institute*, vol. lxxviii. (No. III., 1908).

by the number of tons of pig-iron made in the United States in 1906, represents a saving (assuming 1.75 tons of coal as required to make 1 ton of coke) of approximately 30,000,000 tons of coal.

The progress along this line in blast-furnace practice has been steady and wonderful, and has culminated in the ingenious device of James Gayley, which still further economizes fuel, by freezing the blast before admitting it to the stove, in order to eliminate moisture, and thus supply the stack with a gaseous element of as constant and reliable a composition as the solid elements of fuel and ore.

The advances in blast-furnace practice in the direction of fuel-saving have been great. But they are not as startling or as picturesque as the economies which followed the introduction of the pneumatic method as applied through the mechanical and metallurgical skill of Bessemer, and as developed in the United States through the genius of Holley. We can all recollect the distressing sight, especially in summer weather, of the puddler, stripped to his waist, toiling over his furnace, while burning up from 20 to 27 cwt. of coal, in converting 1 ton of pig-iron into puddle-bar. Leaving out of the question the fuel used in generating the power for operating the Bessemer converters, which, however, is generally recovered from the waste heat of the blast-furnace, the amount of coal saved in making Bessemer steel instead of wrought-iron during the same year of 1906 exceeded 22,000,000 tons.

The metallurgy of copper has benefited as acutely as the metallurgy of iron and steel from the combined science and skill of the mechanical and metallurgical engineers. One recollects distinctly how, in the old brick furnace, a campaign of 10 days, with a daily charge of 10 tons of ore, was looked upon as almost phenomenal; and that from the time we began roasting sulphide ore in heaps until the refined copper was turned out after endless handlings of the mattes, as they were worked up from lower to higher grades, about three months was occupied. Now, by means of mechanical roasting-furnaces, large jacketed cupolas, electrical cranes, the Bessemer converter, and the Walker casting-table, the ore is turned into metal in fewer hours than it formerly took weeks, and at the same time almost dispensing with hand-labor.

While these industrial changes were going on in the mining and metallurgical fields, the electrical engineer was bringing under control that tremendous force which Faraday investigated as dynamic electricity; and we metallurgists have not been slow to apply it, both to the saving of fuel and other natural resources, and to the conservation of human labor. The modern rolling-mill, in which a motor replaces the small engine and boiler that used to operate the rolls, and the modern electrolytic plant which turns out electrically pure copper, are only the more visible benefits that electricity is conferring. When some of us commenced our technical experience, the deduction in precious metals made by the refiner of copper before any contribution was made to the miner or the seller, was $60 worth per ton of ore or metal. Under such heavy charges comparatively small amounts of gold or silver were or could be saved. To-day, through the application of electrolysis to the metallurgy of copper, about $8,000,000 in value, which was formerly lost, is now recovered annually and goes into commerce as a by-product; for the world's copper may be assumed to carry an average of $10 per ton in gold and silver.

The last application of this mysterious force, by transmitting from stationary engines electric current for the movement of trains, aims at reducing what is certainly one of the most wasteful uses of coal—its consumption in the locomotive for the generation of steam. In distributing our coal-supply, the railroad burns up from 20 to 25 per cent. of the total production of our coal-mines. This will be notably reduced, though to what extent has not yet been determined. But before this desirable consummation is attained, if electrical engineers continue to extend the limits within which long-distance transmission can be applied economically, they will bring the latent, neglected forces of the whole continent to our doors, and the water-powers a thousand miles away, as well as the winds and tides, will propel our railroad-cars as well as heat our houses. The service which coal now performs will be fulfilled without the expenditure of human labor and the diffusion of so much obnoxious smoke and vapor. Long before our coal-supplies are exhausted, even on the most pessimistic calculation, our children will gladly leave the balance in the ground, and

charge off to profit and loss some of what we now consider our most valuable natural asset.

There is no doubt whatever that the destruction of our forests is attended by a host of such terrible consequences that a halt must be called. In the early days at Bisbee, when we were at a distance from the railroad, we of necessity almost stripped the hills of their scanty clothing of stunted wood, for we were forced to use wood for the generation of steam. I find from one of the earliest statements that the company burnt about 4,000 cords of wood for the year. The hills for miles around were completely denuded, with the result that disastrous floods have ever since almost annually deluged and damaged the town, which is built in the troughs of two converging valleys. As mining engineers we are sensible of the ruin which reckless lumbering involves, and we lower with regret every stick of timber that we bury underground. Nor are we satisfied to bemoan the fact without making some effort to remedy the evil. It has been suggested, and we are trying the experiment, to replace wood by iron. The forests can be restored in time by reforestration, but iron-ores cannot be replaced. And, therefore, it is a false economy to attempt to save a reproductive material by substituting one which rusts and cannot be regenerated. Concrete is also being used more and more in mining-operations, and against its substitution for wood there can be no objection; but the most notable economy will result from improved methods of mining, especially from the introduction of the caving and slicing systems. These were introduced into the Cananea mines when Arthur S. Dwight was manager; and Dr. Ricketts and Mr. Kirk have extended the use of the methods and applied them so successfully that less than half the timber is used per ton of ore extracted today than was buried in the mine three years ago. The following data, kindly supplied by Dr. Ricketts, represent the saving which is going on at Cananea, and in many mines where the same method is applicable:

Timber Consumed Per Ton (Wet) of Ore Produced at Cananea, Mex.

Period.	Tons Ore Mined.	Feet Timber Used.	Feet Per Ton.
Aug. 1, '05, to Jan. 31, '06, ...	463,039	10,774,342	23.27
Feb. 1, '07, to July 31, '07, ...	554,473	8,268,682	14.95
Aug. 1, '08, to Sept. 30, '08, ...	97,510	1,091,837	11.30

While it would be presumptuous to pretend that, as a people, we are economical, and to deny that, under modern corporate control of large national resources, the temptation, under necessity of making large profits, is not betimes stronger than the appeals which conscience makes to subordinate personal gain to the national welfare, I am sure that neither our largest mining and metallurgical companies nor ourselves, as their working-agents, are recklessly indifferent to the preservation of those very materials upon which the wealth of the corporations and our own salaries depend. No large corporation would to-day use an old boiler and slide-valve engine with a consumption of 6 lb. of fuel per horse-power-hour in preference to a triple-expansion, cut-off engine which will do the same work with 1.5 lb. per horse-power-hour, and so on through the whole gamut of operations which these large corporations conduct and which we, as their managers, advise them to adopt, because we believe them to be the best and most economical methods.

While public policy may not be the prime motive for saving, every thinking man in a large institution, from the manager downward, takes a pride in knowing that he is saving, and feels a sense of shame when he is conscious of wasting. And in economic life—I do not speak of social and domestic life—the rules against waste are becoming more and more rigid and are better enforced. The public outcry, therefore, against the large corporations for wasting the natural resources of the nation is unjust in so far as it fails to recognize what they have done and are doing in the direction of conservation, and inasmuch as it gives the working-staff of these great corporations so little credit for the marvelous progress the world has made through their instrumentality. They have saved where formerly, through ignorance and inexperience, their predecessors were wasting. With more profound knowledge, and better instruments for observation and investigation, they are patiently unraveling nature's secrets and learning how to turn her forces to human uses. I cited a case of the unavoidable waste of copper-ore, of fuel and of human labor in the treatment of the oxidized copper-ores of Arizona 20 years ago. The men who were wasting acted upon their knowledge and skill. So now it often happens that in response to the urgent call which modern

society makes by fits and starts for enormously increased productiveness of various commodities, the demand can be met only at the expense of waste of nature's resources, of human energy, and even of human life. If a more staple balance could be maintained between supply and demand; if the current of domestic and economic life would run more smoothly; if wealth were not accumulated so easily and spent so lavishly; if those marvelous improvements to which we have referred were not periodically made, which give these irresistible impulses to world-wide human energy, thereby bringing about these oscillations between hard times and good times, between labor-dearth and labor-surplus;—if all these disturbing elements were obliterated, certainly there would be less waste, and possibly there would be more happiness. But it is neither our part nor within our power, as mining and metallurgical engineers, to reconstruct society or renovate the world. Yet it is our duty to continue using our best efforts—whether the world recognizes our merits or not—to get the utmost energy out of human life as well as out of the inert material we handle—with the least possible exhaustion of human tissue and the smallest possible waste of mineral or vegetable material.

Driving Headings in Rock Tunnels.

BY W. L. SAUNDERS, NEW YORK, N. Y.

(New Haven Meeting, February, 1909.)

THIS paper deals specifically with heading-driving as distinguished from the broader term tunnel-driving. A heading is a pilot or path-finder for the main tunnel. Some headings are complete tunnels in themselves; that is, conditions at times warrant driving a heading the full diameter of the proposed tunnel. A tunnel 10 ft. in diameter might be driven to advantage through a single heading, provided the nature of the material admits, but it usually pays to drive rock tunnels of large diameter through the heading-and-bench system. Much depends upon the material, as in soft ground tunnels of much larger diameter are driven in a single heading.

In mines, the tunnels, drifts, cross-cuts, adits, etc., are usually of small diameter, hence the driving of headings applies to completed tunnels in mining-work, except, of course, in cases like drainage-tunnels and main entries leading into coal-mines, where conditions call for tunnels of large diameter, approximating those driven for railway-service. In building aqueducts the tunnels are driven from 6 to 8 ft. in diameter up to 15 or 20 ft., while in railway-service the dimensions vary from 18 to 30 ft. in diameter.

There is no department of rock-excavation so difficult as that of heading-driving. This work also belongs to the most expensive class in rock-excavation. A heading is driven directly in the solid. There are no lines of lesser resistance towards which to direct the energy of the blast, but the material must be blown out by main force. The completion of the heading simplifies all the rest of the work of excavation. It is easy, for instance, to enlarge a heading either from above, below or on the sides, by breaking towards the open face through holes approximately at right angles to the axis of the heading; or the enlargement sometimes takes place by holes driven outside of the heading running approximately parallel with it, the rock

being broken into the heading. All of this is bench-work, or stoping; it involves little difficulty, and a minimum of explosive is required, because there is an open end or face towards which the energy of the blast is directed.

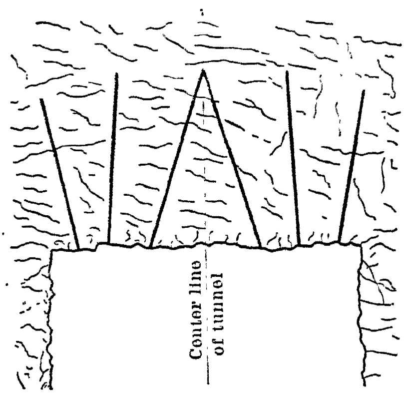


FIG. 1.—PLAN SHOWING DIRECTION OF HOLES IN TUNNEL-HEADING.

Fig. 1 shows the usual arrangement of heading-holes in tunnel-driving. After blasting the center-cut holes and the first of the side-round holes, the appearance of the heading in elevation will be about as shown in Fig. 2. The latter illustration

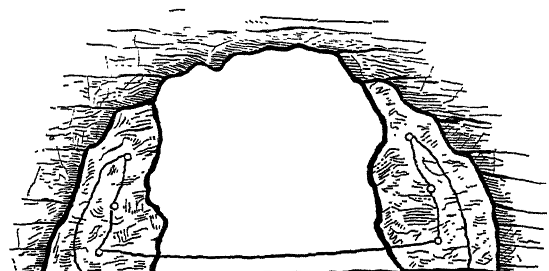


FIG. 2.—METHOD OF SHOOTING SIDE-CUT HOLES IN TUNNEL-HEADING.

shows the second set of side-round holes connected up by wires ready for firing.

Fig. 3 shows in plan and elevation typical conditions where tunnel-headings are to be enlarged. This is stoping-work, and because of the advantageous position which the holes can be given, the removal of the rock may be accomplished at much less expense per cubic yard than in a case of heading-driving.

Driving seems to be the fitting term to use for tunnel-construction because, from the beginning to the end, every other consideration is sacrificed to progress. The tunnel is worthless until completed, and the capital involved in its construction is locked up and unprofitable. During the construction of the Simplon tunnel in the Alps premiums were offered exceeding $1,000,000 for increased speed in driving. This

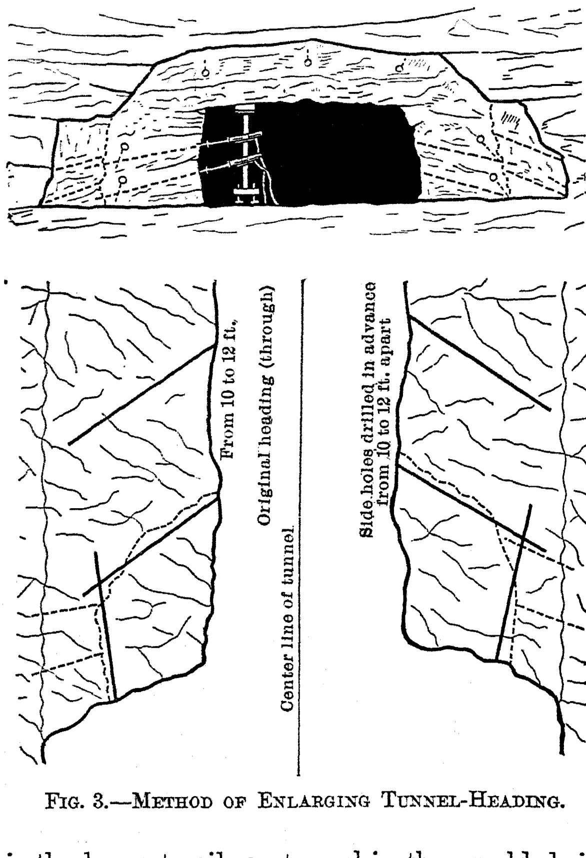


FIG. 3.—METHOD OF ENLARGING TUNNEL-HEADING.

tunnel is the longest railway-tunnel in the world, being 12.25 miles long, and it holds the record for speed in heading-progress.

Progress in heading-driving is dependent, first, upon an efficient boring-apparatus and system, and following closely upon this, and of perhaps equal importance, is the system of blasting. The nature of the rock has a great deal to do with heading-progress. A hard rock which shatters easily is usually

more favorable than a tough, soft material. The direction of the dip or of the cleavage of the rock has a good deal to do with progress. Fissures, pressure in the rock, water-bearing strata, heat, gas, and soft spots, resulting in falls, are hindrances to progress.

Driving a heading in solid rock is, of course, primarily a job for the rock-drill. Where the best records are made the natural inquiry is as to drill employed, yet the importance of this is frequently overestimated. The system is of equal importance. The best rock-drill, or the one which drills fastest, might be handicapped if insufficiently mounted, since the holes driven in headings are invariably of moderate depths, so that much of the time is employed in changing the drill from one hole to another, changing steels, etc.

Too little importance is given to the question of how a rock-drill is mounted. An illustration of this is shown by comparing the drilling-capacity of a machine when used, for instance, on a tripod or a column with that of the same type and size of rock-drill mounted on a gadder-frame or quarry-bar. The usual work of a rock-drill in average material is from 50 to 100 lin. ft. of hole in a day of 10 hr. This machine, when mounted on a gadder-frame or quarry-bar, will do 350 lin. ft. of hole in the same time, the difference being made up, not in the drilling-capacity, but in the facilities afforded for rapidly changing from one hole to another.

The remarkable records in tunnel-driving made by the hydraulic drills used at the Simplon tunnel, referred to later on in this paper, have always been attributed to the efficiency of the boring-machine. These Simplon headings were driven at the rate of about 20 lin. ft. in 24 hr., and in hard rock, while American tunnel-driving seldom exceeds 10 ft. in 24 hr. when driving in hard rock, so that here we have a difference of 2 to 1 against the American system. It is a common thing to say that a single-track railway-tunnel driven from two headings will progress at the rate of a mile a year. This is our average progress under average conditions. In the Alps the tunnels are driven from two headings at the rate of about 2 miles a year. These data are, of course, general, and do not take into consideration delays due to falls, destructive pressure in the rock, hot-water streams, etc.

It will be seen later on that the remarkable records made at the Simplon have practically been equaled by the work now going on in the Loetschberg tunnel, this latter being equipped with American percussive drills mounted on carriages similar to those used at the Simplon.

It may safely be said that the discovery of gunpowder, followed by that of dynamite, is the greatest invention that has been given us in facilitating tunnel-driving. Next comes the power-drill. In ancient times rock-excavation under ground was limited to hand-tools, wedges, etc., assisted by a system of excavation known as "fire-setting," which consisted in heating the rock and suddenly cooling it with water, thus disintegrating it. Pliny mentions this fire-setting system, and we are told that Hannibal used it to disintegrate the rocks while crossing the Alps.

The ancient Greeks and Romans were expert engineers in tunnel-driving. Herodotus mentions a tunnel in the island of Samos cut through a mountain 900 ft. high. Its length was 4,248 ft. and its cross-section 8 by 8 ft. This tunnel was built during the sixth century B.C. We may well be astonished in studying the records of tunnel-driving by the Romans. They built tunnels for drainage, for passages, aqueducts, etc., through rock and through earth, not only in Italy, but throughout their possessions. The drainage-tunnels built by the Etruscans, from whom the Romans learned the art, are among the greatest engineering achievements of antiquity. A tunnel in use at the present day was built through the Apennines between Naples and Pozzuoli in the first century B.C. This tunnel is said to have been originally 0.75 mile in length; height, 30 ft.; width, 25 ft. A complete history of tunnel-driving "from the reign of Rameses II. to the present time" is given in Drinker's classic work.[1]

American Tunneling-Records.

The following are some of the best American tunneling-records,[2] arranged progressively. In some cases the records are those of complete tunnels and in others of the heading only,

[1] *Tunneling, Explosive Compounds, and Rock Drills* (1878).

[2] *Engineering News*, vol. lix., No. 18, p. 377 (April 2, 1908); vol. lx., No. 1, p. 9 (July 2, 1908); No. 4, p. 102 (July 23, 1908); No. 21, p. 570 (Nov. 19, 1908).

but as a heading is in itself a tunnel they may all be considered in the same class as records of progress. The figures represent monthly progress:

Musconetcong, N. J., 1872,	144 ft.
Nesquehoning, Pa., 1871,	165 ft.
Hoosac, Mass., 1865–1873,	184 ft.
Busk, Colo., 1890–1893,	202.5 ft.
Stampede, Wash., 1886–1888,	274 ft.
Cascade, Wash., 1897–1900,	301 ft. [a]
Aspen, Wyo., 1901,	306 ft.
Bitter Root Mts., 1908,	340 ft.
Kellogg, Idaho, 1898,	345 ft.
Raton, Colo., 1907,	412 ft.
Sutro, Nev., 1869–1877,	417 ft.
New Croton Aqueduct, 1887,	127 ft. in one week.
Gunnison, Colo., 1908,	449 ft.
Elizabeth Lake, Colo., 1908,	466 ft.

[a] Another record for this tunnel is: completed tunnel, both ends, Nov., 1899, 527 ft.

There is some question as to whether the Sutro record is for one heading or two, so that its place in the list is questionable; also, notwithstanding the mental allowance for the differences in working-conditions, the Raton tunnel can hardly be considered as in the same class with the others, as the material, not to call it rock, was so soft that most of the drilling was done with coal-augers, and a steam-shovel was used to excavate the bench. The tunnel was finally lined with concrete 2 ft. thick.

The record of the new Croton Aqueduct, in 1887, of 127 ft. in one week, was a deliberate drive for one week for the purpose of making a record. This was done regardless of expense or the amount of explosive consumed. I was on the ground and know that while the figures represent the true progress made, yet they are reliable only in that they show what may possibly be done, and not what ought to be done or what can be done, under similar circumstances. The week following that during which the record was made the progress fell off about one-half.

The record of the Bitter Root Mts. tunnel, driven by Winston Bros. Co., "is probably the record rate of progress on American railway-tunnels driven the full width of the arch in hard rock." This tunnel was driven by the regular top-heading

system. The section of the tunnel is rectangular with semicircular roof-arch. The width is 21.33 ft.; height to springing, 15.25 ft.; height to crown, 25.92 ft.; rock, quartzite, slightly laminated. In the six months beginning with June, 1908,[3] the averages were: East heading, 289.3 ft. per month; west heading, 281.2 ft.; both headings, 570.5 ft.; bench, both ends, 632.5 ft. The average progress, both ends, for the first 11 months of 1908 was 537.6 ft. per month.

This tunnel is timbered throughout except for 1,302 ft. of the west end. The section is 18 ft. 6 in. by 25 ft. in the clear, or inside the timber. Perhaps the most gratifying feature of the work is the increase of speed as the work progressed, as shown below:

Heading,	one end, . . .	June, 333 feet.	Nov., 340 feet.
	both ends, . .	Aug., 628 feet.	Nov., 608 feet.
Bench,	one end, . . .	Aug., 415 feet.	Nov., 527 feet.
	both ends, . .	June, 634 feet.	Nov., 855 feet.

In the west heading no advance was made for the last six days of November, as a seam of very wet running ground, talc, was struck, necessitating a change of arrangements. If the rate had been maintained for these six days the total would have been 674 instead of 608 ft., thus exceeding the previous record.

As is to be always remembered, each of these tunnels differs from all the rest as to the conditions, favorable or otherwise, so that inferences from the comparing of records are not necessarily authoritative or final.

It happens that the last two on the list, the Gunnison and Elizabeth Lake tunnels, are probably as fairly comparable with each other as any, with the interesting particular that the two were driven by different and contrasting methods. Both are in granite and about 12 by 12 ft. in section. The Gunnison tunnel was made by driving the section one-half size; that is, by a smaller heading 6 by 12 ft. The Elizabeth Lake tunnel was driven full size by the lower-heading method.

European Tunneling-Records.

Following our list of American high records we have now a few representing the best European practice. These are

[3] Private communication to the author.

arranged progressively, and after the first two or three items this list would tag on to the other, showing an increasing rate from beginning to end of the combined record, only that, unfortunately, these latter records generally antedate or are contemporaneous with the American. The highest monthly records of European tunneling show not only larger figures but a better maintenance of approximately the maximum, month after month. In some of the cases here following several successive monthly records are given:

Mont Cenis, 1857–1870; 297 ft.

St. Gothard, 1872–1881; 436 ft., with a year's average of 343 ft. (This is the only Alpine tunnel driven with top heading.)

Ricken, 1903——; 452, 461, 413, 358 ft. (Hand-drilling entirely. Work was stopped on this tunnel nearly a year on account of fire-damp.)

Bosruck, 1902–1905; 546, 526 ft.

Karawanken, 1902–1905; 552, 544, 553 ft.

Arlberg, 1880–1883; east heading, 556, 594, 610, 613, 637 ft. (Percussion-drills.)

Arlberg; west heading, 509, 527, 625, 641 ft. (Hydraulic rotary drills.)

Albula, 1900–1902; 558, 607 ft. (Hydraulic rotary drills.)

Tauern; 548 ft., with an average of nearly 525 ft.

Loetschberg; 555, 574, 538, 558, 551, 592 ft.

Simplon, 1900–1905; north heading, 682.2 ft.

Simplon; south heading, 685.5 ft. (This is the world's record.)

Seven or eight of the best records are so nearly alike that the differences might easily be accounted for by the varying hardness of the rock or other material conditions, without implying anything as to the superiority of the apparatus or the system employed in either case.

The Simplon and the Loetschberg tunnels are perhaps the most interesting of the list, and they are quite intimately related to each other. The Loetschberg tunnel is destined to connect directly Berne, Switzerland, with the Simplon tunnel, thus establishing a direct communication between Italy and Alsace-Lorraine, Belgium, Holland, and the Rhine provinces, greatly shortening the trip between Berne and Brigue. The total length of the tunnel will be 8.5 miles, and it was being driven from both ends—Kandersteg at the north end and Goppenstein at the south.

Fig. 4 is a map showing the location of the Loetschberg tunnel in relation to the Simplon tunnel.

As we know, this tunnel has had troubles of its own, work at the north end being stopped absolutely after a frightful and

unusual accident. Though the line passed under the Kander river, there was 600 ft. of covering over the tunnel at this point, 300 ft. of this being known to be bad, but the engineers seem to have taken the chances on the other 300 ft., and the chances proved to be deadly certainties against them. The entire depth caved in, forming a funnel-shaped hole from the bottom of the river, killing 26 men, causing a loss of all the machinery, and leaving a problem in tunnel-work such as, perhaps, was never encountered before.

The work at the southern end of the tunnel from Goppen-

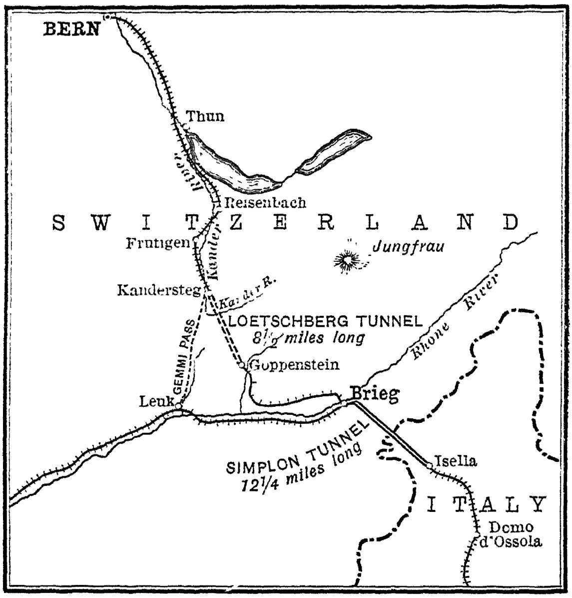


FIG. 4.—MAP SHOWING LOCATION OF LOETSCHBERG TUNNEL.

stein, Valais, Switzerland, is still going on at full speed, there having been an advance of nearly 2.5 miles up to the present time. The altitude of the work is about 4,000 ft. above sea-level. The tunnel is built for double tracks, with an area of cross-section of 592.5 sq. feet.

What is known as the Belgian method is employed in driving the tunnel, there being one principal bottom heading of 60 sq. ft. and an upper heading of 35 sq. ft. Every 600 ft. upraises are made from this bottom heading, and a top heading is started from each of these upraises. From the top heading the work of taking out the tunnel to its full section is carried

on in bench-work. Besides the upraises, chutes about 2 ft. square are blasted out between the bottom heading and the top, where the muck can be dropped into trains of cars below. To make sure that there will be no dropping of the middle portion, this is supported by timbers as long as necessary, the timbering being successively removed and carried forward. The sides are finally trimmed out, and a concrete lining is placed.

Fig. 5 is a longitudinal section of the Loetschberg tunnel, showing the upper and lower headings, upraises, chutes, etc., with drills in position; a transverse section of the tunnel, showing the position and size of the top and bottom headings in relation to the completed tunnel; and an enlarged outline-elevation of the tunnel-carriage.

The responsibility for the total advance thus always remains with the bottom heading, and here is the most interesting feature of the work. There is a single track, about 30-in. gauge, in the bottom heading, this heading being widened out where the upraises are to come, and the track being turned out at these points. On this track travels a special car, which carries a normally horizontal, tilting, counterbalanced steel beam, upon the forward end of which is secured a horizontal bar, and upon this bar are four drills. The swinging of the beam upward or downward, and the placing of the drills above or below the bar, gives them all the positions for the 12 or 14 holes usually required for each round.

The original car and mounting designed for this service may be described as too much of a good thing, and it was never used. On this car were two of the balanced, vertically-swinging beams, each carrying its shaft or tunnel-bar, and on each of these three or four drills would have been mounted. The ends of these two bars were telescoped, and it was designed to clamp and hold the bars in working position by forcing the ends out against the sides of the heading by hydraulic pressure. An important and conspicuous feature of the car was, therefore, an air-operated duplex water-pump to furnish and maintain the required pressure.

The car actually used was a much simpler affair, and of this I have only a photograph taken from the rear, with the apparatus in working-position in the heading. Fig. 6 is a view of

the lower heading in the Loetschberg tunnel with the drills at work, showing the weight on the rear end of the tunnel-carriage. This car was built by the Ingersoll-Rand Co. It carries only a single balanced, vertically-swinging beam with a single bar pivoted to swing horizontally above the forward end of it, and the bar is secured against the sides of the heading by the usual jack-screws, so that the water-pump is dispensed with. The bar is pivoted above the beam so that when the car is run backward or forward the bar may be swung around, above and parallel with the beam, leaving no interfering projections to catch the sides of the heading. Four $3\frac{5}{8}$-in. Ingersoll-Rand drills are mounted on the bar, each, of course, with a separate air-connection, but all connecting by a manifold with a single hose at the rear of the car.

Either 12 or 14 holes are drilled in each round. These are 4 ft. deep and 2 in. at the bottom, in four vertical rows, the inner rows running nearly parallel with the tunnel axis, the same as the rest. As soon as the holes are all drilled the bar is swung around straight and the car is run back to the last turnout until the blasting is done. Before the blasting, a $\frac{3}{8}$-in. steel plate, 6 ft. 6 in. by 3 ft. 3 in. in area, is laid down just ahead of the end of the track, and after the blasting is over a cut is quickly made through the center of the muck-pile down to the plate, and the tunnel-carriage is run forward on this, the bar is set again, and the drills begin the top row of holes. Mucking-out continues during the drilling.

The use of this carriage facilitates the mucking to a considerable degree. After the blast in the usual American tunnel there is a mass of muck reaching nearly to the roof piled up in front and close to the face of the heading. This muck is shoveled away only to a sufficient degree to allow the men to climb over it and dig holes close to the face for the purpose of placing the columns which carry the rock-drills. The use of this carriage, with the bar which carries the drills projecting forward some 12 ft. or more from the truck, requires that the first mucking be done on the steel plate on the floor, and only sufficient to make a trench through the center of the heading, so that the tunnel-carriage may reach close enough to enable the drills to begin operations in the top holes over the muck. Here we have the drills at work earlier than with the

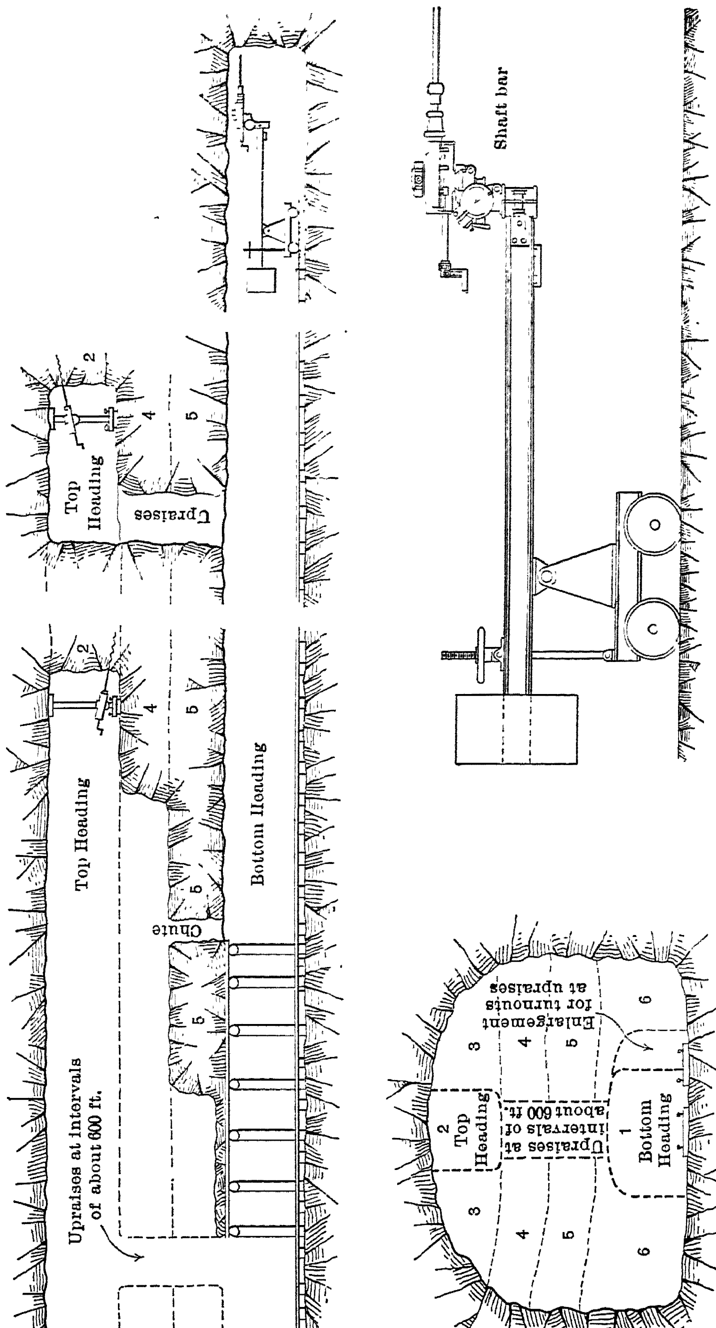


FIG. 5.—LONGITUDINAL AND TRANSVERSE SECTIONS OF LOETSCHBERG TUNNEL, SHOWING UPPER AND LOWER HEADINGS, UPRAISES, CHUTES, ETC., AND DRILLS IN POSITION, WITH OUTLINE-ELEVATION OF CARRIAGE AS ACTUALLY USED IN THE HEADING.

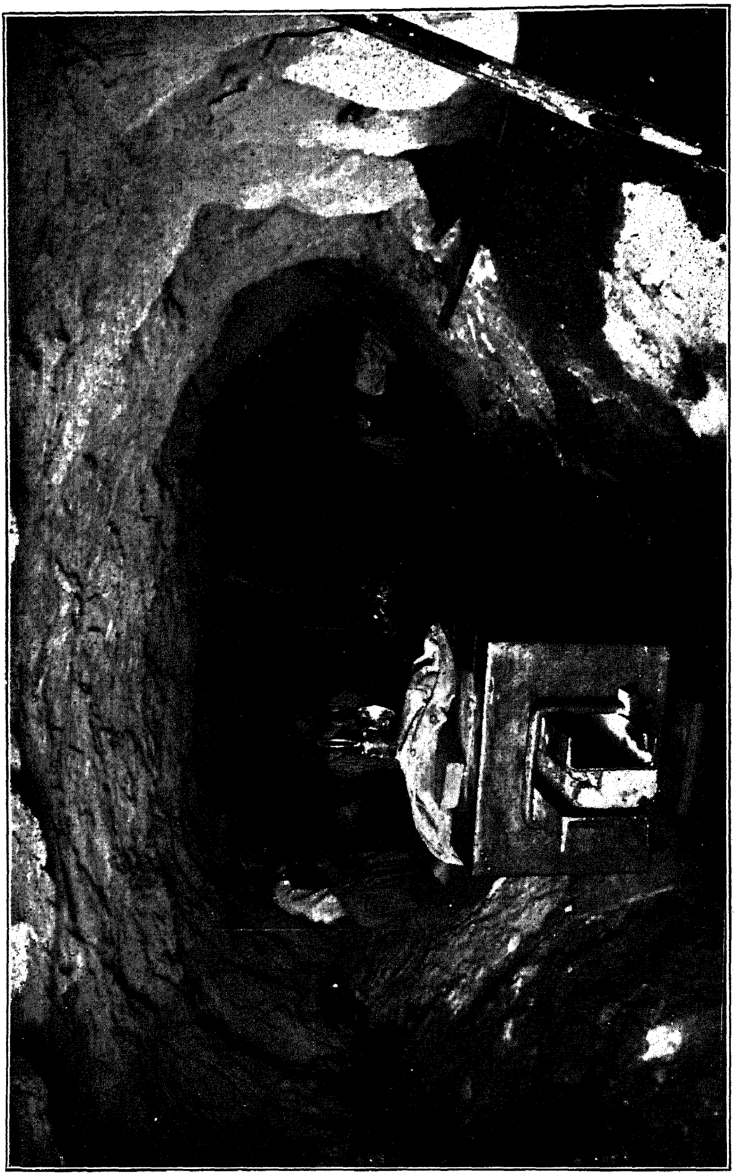


FIG. 6.—HEADING WITH DRILLS AT WORK, SHOWING REAR OF CARRIAGE.

FIG. 7.—ROCK-DRILLS IN HEADING OF COLLEGE HILL TUNNEL, PROVIDENCE, R. I.

FIG. 9.—THE RADIALAX CHANNELER CUTTING A CHANNEL IN TUNNEL-HEADING.

American system, and the muckers have a better opportunity to load the cars, because the material is scattered on each side of the heading instead of being piled up only at the face.

Where the rock-drills are mounted on a bar carried by a tunnel-carriage it is easier to keep them in condition, free from muck and grit, than with the American system, where they are detached from the column-arms and laid on the floor. This is an important point, not only in reducing maintenance-expense and decreasing wear in the cylinder and other moving parts, but also in lessening the difficulty in keeping the stuffing-boxes tight, resulting in a smaller leakage of air.

Blasting is done by fuse-and-cap method, the fuses being all fired at the same time, but the length of the fuses is such that the center holes are fired first. By this system time is saved; but there is some danger of missed holes, and to minimize this three fuses are used in the bottom holes and two in all the others. The explosive used is 60-per cent. dynamite, made at Brieg. The center holes are charged with 2.7 kg. (6 lb.) each, and the average total charge for the 12 or 14 holes is from 24 to 26 kg. (53 to 57 lb.). The holes are not very deep, not exceeding 4 ft., starting 2.5 in. and finishing 2 in., to take cartridges up to 50 mm., or nearly 2 in., in diameter.

To accomplish the rapid rate of advance which is maintained the number of men employed is large. Night is, of course, the same as day, and the work is continuous. There are three 8-hr. shifts, and each shift is expected to drill two rounds and to shoot twice, making about 7 ft. per shift, or from 18 to 24 ft. per day. A liberal bonus is paid to the men all around—Italians from the Northern provinces—for all speed above a certain rate. Each separate drill has 2 men, and 2 additional helpers handle the steels, 10 men clear away after the shot, and all the 20 men work together for the placing of the truck and the laying of its side track, located about every 600 feet.

Experience has shown that the truck is indispensable if the rapid advance is to be maintained, though it is not without its objectionable features. The fact that all the four drills on the bar are in the same horizontal plane makes it difficult to give the upwardly-inclined holes just the same angle they could have if the drills were on separate columns, and to get a clean break to the bottom of the hole more explosive must be used.

This disadvantage in a horizontal bar over the vertical columns is more than compensated for by the readiness with which the bar may be set in place and jacked. One bar only has to be jacked, and that across the tunnel horizontally against the walls, which are usually rigid, while with the column system there are two, sometimes three, columns to be placed in position, each in a separate place, and each one must have a firm base, which is not always available close to the face of a heading just after the blast. But this is not all, nor is this the most imporant function of the tunnel-carriage system. By its use, and because of the facility which it affords for readily mounting

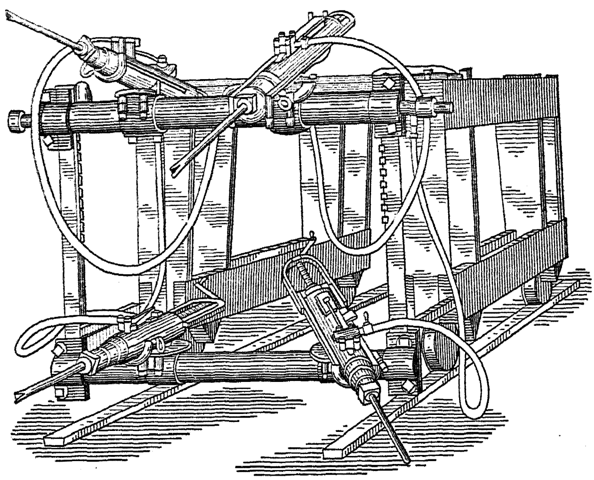


FIG. 8.—TUNNEL-CARRIAGE WITH OLD BURLEIGH DRILLS.

the rock-drill and moving it from one hole to another, it is possible to put in a large number of shallow holes of large diameter, as against the American system, which is a small number of deep holes of small diameter. It must be understood that when a rock-drill mounted upon a column has once begun its work in a heading, so much time has been occupied in placing it in position, getting the jack-screws tight, and pointing the hole itself, that we have fallen into the habit of keeping it there and putting in as deep a hole as we reasonably can consistent with the dimensions of the heading, the nature of the rock, and the angle of the hole.

Fig. 7 is a view of the rock-drills at work in the heading of the College Hill tunnel, Providence, R. I. This is a practical

working scene, showing a common method of driving a tunnel of small diameter, one column being used with two arms, with one drill on each arm.

Other things being equal, a deep hole can be drilled faster than two or more shallow holes, the combined depths of which are equal to the deep hole. The difference is made up in the time lost in changing the mounting, whether that mounting be a column or a tripod. The deeper the holes are driven the smaller is the diameter of the hole at the bottom, and we know, of course, that it is at the bottom of the holes that we want the greater amount of concentrated explosive force, and that the larger the hole is at the bottom the better is the effect of the blast.

A theoretically-perfect hole would be one larger at the bottom than at the orifice; but this is not practical in tunnel-work, hence we must suffer the loss of gauge, and by an excessive amount of explosive, by good tamping, and other means, try to make up for this handicap.

Another habit which we have fallen into is, that too small a number of holes are inserted in the heading. This is because we have found it difficult to make reasonable progress in hard rock by drilling an adequate number of holes, owing mainly to the fact that the mounting must be changed from one place to another. Having the habit of deep holes, we find it necessary to start them in a certain particular place, so that they will bottom just where we want them in order to "break the ground." Now, it is just in this that we see the disadvantage under which the column system suffers.

Having the horizontal bar used in the Loetschberg tunnel with four percussive drills mounted upon it, and this bar being practically balanced on a little tunnel-carriage and easily jacked in place across the walls, we begin putting in a hole with each machine. If one bottoms its hole ahead of the others it is simply swung on its radius, the center of the axis being the bar itself, and another hole put in. All of these holes are shallow, and being shallow they are naturally of large diameter. One meter is the usual depth, with a maximum of 4 ft. This is from one-half to one-third the usual depth of hole inserted in a tunnel-heading under the American column system; and because the holes are shallow it is not necessary to start

them in so exact a position, but as long as we have enough of them, and use plenty of dynamite, the rock is thrown out in pieces of smaller size, and usually farther away from the face of the heading, than where the blast is from a deep hole, even though it may be directed to meet another hole, the two holes practically joining at the bottom, forming a wedge, and acting to concentrate the force by means of which the center cut is made.

This tunnel-carriage is nothing more or less than what is shown in Fig. 5, and is so simple and so heavy that it is practically indestructible.

What we have always understood as a tunnel-carriage is a cumbersome affair, quite different from that used in the Loetschberg tunnel. A carriage was the first form of mounting adopted when rock-drills were used to drive the Hoosac and other American tunnels. The carriage usually occupied the entire area of the heading, and was a hindrance to progress rather than otherwise.

Fig. 8 shows the Burleigh drill-carriage used in the Hoosac and other early tunnels. This is the original form of tunnel-carriage, and attention is drawn to a comparison between the Burleigh carriage and that used in the Loetschberg tunnel, shown in Fig. 5. The Burleigh carriage occupied too much space and was too cumbersome for practical tunnel-driving. It did not afford facilities for mucking, and too much delay occurred after a blast before the carriage could be run close enough to the heading for the drills to begin work. The Loetschberg machine, with its small truck and short wheel-base, need not approach very close to the heading because of its overhanging arm. The two to four drills, mounted on the single shaft carried by the arm, are quite sufficient for all ordinary heading-purposes.

The Alpine tunnel-carriage is a little truck with a wheel-base of only about 4 ft. and a gauge on the track which corresponds to the regular track used in the heading for conveying the material. It consists of a pair of axles with four wheels, and a cast-iron body with a central support, on which the I-beam carrying the shaft-bar is pivoted. On the opposite end of this I-beam is a heavy weight by which the bar is balanced, and by means of a vertical screw and a nut operated by a hand-wheel this I-beam is see-sawed to any position desired.

Next in importance to the system of mounting the drills is the system of blasting. Because of the larger number of holes, the greater diameter, and because the holes are not directed on lines of maximum breaking-efficiency, a great deal more explosive is used than in the American system, but dynamite is cheap—much cheaper in fact than time and labor. To blast by fuse instead of by electric battery would seem to be a step backward, and yet in this class of work it has some advantages. With the electric system the heading is wired and the center or cut holes blasted first. After this the wires leading to the side rounds are connected, and it frequently happens that the first blast has damaged the wires or has covered them so that considerable time is lost in getting ready for the side-round blast. Broken wires result in mis-fires. The use of the time-fuses means that the whole heading is fired practically in one operation, though there is a lapse of a few seconds between the discharges, owing to a difference in the lengths of the fuses, the shorter fuses being connected with the center-cut holes, and the length of fuse being increased in proportion as the side-rounds are blasted from the center. Mis-fires also occur with the fuse system, but this is minimized by employing two or three fuses and caps in each hole.

It takes less time to connect these holes by fuse and to fire them than it does to connect the wires, to see that they are properly insulated, to couple up the leading wires, and to discharge the battery, especially so when through the system of deep-cut holes it is found necessary in American tunnels to blast the center first and the side rounds alternately afterwards.

Other details of the tunnel-work as a whole need not be considered in the present paper. The upper heading is driven to keep pace with the lower one, but in the individual faces here the rate of advance is slower. For this work two 3.5-in. drills are used on 5.5-in. columns, and of course fewer men are called for. This heading has a sectional area of about 40 sq. ft., and the daily advance of each face is about 10 feet.

A large number of machines and accessories are used for the enlarging of the tunnel—3-in. drills on tripods, and a large number of hammer-drills in finishing off the walls, drilling pop-holes, etc.

The initial motive-power available is an electric current up

to 2,500 h.p., developed from water-power. There are two Ingersoll-Rand air-compressors, with a total capacity of 4,250 cu. ft. of free air per minute. These discharge into four receivers at a pressure of 110 lb. The pipe-line into the tunnel is specified to be of such dimensions that the loss of pressure shall not exceed 2.2 lb. in about 6.5 miles, the maximum distance expected to be driven from this end.

Besides these compressors for driving the drills and smaller work, there are also two 4-stage compressors with a free-air capacity of 920 cu. ft. per minute, delivering at a pressure of 1,700 lb. into receivers made up of 12 tubes with a total capacity of 425 cu. feet.

There are two compressed-air locomotives of from 150 to 200 h.p., capable of hauling very heavy loads up a 3-per cent. grade and giving excellent service. These are of French manufacture, while practically all the other machinery is American.

On the line approaching this tunnel there are about 20 short tunnels, some up to a third of a mile in length, which have been driven in preliminary section for a single temporary track, some by hand and others by the use of the Electric-Air drill. These drills have worked, many of them, in places where it would have been impracticable to supply air, and, independently of this consideration, they have given excellent satisfaction.

Returning to the main tunnel, it will be realized that there has been a thorough study and a careful systemization of the entire work, for the purpose of avoiding delays and interferences, and for keeping the necessarily slower work at as rapid a pace as possible. The series of operations at the heading for the entire round have been tabulated as follows:

	Minutes.
Setting up machines,	20
Drilling 12 to 14 holes,	50
Taking down and moving back,	20
Charging holes and firing,	30
Clearing out smoke,	20
Clearing away muck in central part,	80
Extra for unforeseen delays,	20
Total,	240

These operations follow each other and must wait for each

other, while the others not mentioned, principally the removing of the muck, can go on between times.

Tunneling-Machines.

If we could keep the rock-drills running all the time, and eliminate all the other operations except as they could be carried on without interfering with the drilling, that would certainly seem to promise more rapid progress. This is the idea which the inventors of the tunneling-machines are working on, and it is no wonder that there are a great many struggling to develop something practical along this line. At the same time it must be said that there is not in use at the present time a commercially successful tunneling-machine.

The idea is misleading. With the drilling-and-blasting system, the drills cut considerably less than 1 per cent. of the material; the tunneling-machine proposes to cut up practically all the material into chips, or to do more than a hundred times as much mechanical cutting of the rock with the time multiplied only by three or four, and no allowances for machine- or drill-arrangements or other time-losses.

Were nitro-glycerine and dynamite undiscovered, the tunneling-machine would be of great importance. It would seem fundamental that, in counting the cost, it is more economical to dislodge rock from its strong place in a tunnel-heading by discharging a high explosive at the bottom of a hole than by doing all of this work through the pulverization of the entire mass, especially so as the material removed may be of large or small size, the main point being to get it out of the way. An argument used by the advocates of tunneling-machines is that the chief problem in driving is to get rid of the material. There is a good deal of truth in this when applied to tunnels of large diameter and to headings where the material is discharged *en masse* at the face and in large and irregular pieces. The Alpine system reduces the importance of the problem of getting rid of the material, for the reasons that have been stated.

A tunneling-machine which attempts to bore a hole 2, 3, or 4 ft. in diameter is a reasonable proposition, but the difficulties are usually increased out of all proportion as the diameter of the tunnel is enlarged. It is not uncommon to put in holes with a large rock-drill, using bits 6, 8, and even 12 in. in

diameter. This might be extended to several feet by increasing the diameter of the drill in proportion as the size of the bit is enlarged. The most reasonable of the numerous tunneling-machines that have been suggested is that known as the Karns, described later on, which is nothing more or less than a rock-drill with a large bit. Its limitations are mainly in connection with large diameters and because of the irregular nature of the rock, causing it to strike harder on one portion of the bit than on the other, and the effect of this is, of course, more destructive as the diameter is enlarged.

Radialax System of Tunnel-Driving.

A heading-driving system, known as the Radialax, which has been used in coal-mines, and to a limited extent in tunnels, is that in which a single channel is cut in the face of the heading. This channel may be from 2 to 4 in. in width, and preferably 4 or 5 ft. deep. Having such a cut, preferably in the center of the heading, the problem of removing the surrounding rock becomes a simple one, because the heading is now nothing more than a double bench, and we know that the amount of explosive required in bench-work is considerably less than in headings, hence a minimum of explosive is used and the shock of blasting is considerably reduced. This system has special advantages when driving tunnels under foundations of buildings and through places where the noise and shock from blasting are destructive or objectionable.

Fig. 9 shows the Radialax machine cutting a channel in the face of a tunnel-heading. This machine consists of a modified form of rock-drill mounted upon an arm which is, in turn, mounted upon a column. A common X-bit is used, and, while it is being reciprocated by the drill, the operator, by means of a worm and quadrant, swings the bit radially, thus cutting a vertical channel. Owing to this radial movement the channel cut is longer at the bottom than at the face of the rock. The channel need not reach from roof to floor in order to be effective as a center release-line towards which the side-round blasts are directed.

The tunneling-machines here mentioned are only typical of a great number. H. A. Everest, in a graduating-thesis at the Colorado School of Mines last year, gave a record,[4] beginning in

[4] *Mining and Metallurgical Journal*, vol. xvi., No. 21, p. 4 (Sept. 5, 1908).

1853, and mentioning about 30 different machines. The brief descriptions of three or four machines here given are partly abstracted from that paper. The inventors of these machines recognize the most urgent problem of tunnel-driving: how to advance the heading more rapidly. That none of them will achieve a practical solution which may ultimately materially lower the time-record for tunneling in hard rock it is perhaps unsafe to predict.

Proctor Tunneling-Machine.

This machine is the joint production of O. S. Proctor, of Denver, and E. F. Terry, of Terry & Tench Co., contractors, New York. This is to cut a tunnel or heading 8 ft. in diameter, converting all the material into dust or chips. There is a four-wheel truck at the rear which travels on a track, and the front end is carried by two conical wheels, which fit the circle of the tunnel on each side and thus keep the head central. The main shaft, which is hollow, has adjusting-screws at the back by which the direction of advance may be controlled either vertically or horizontally. In the middle of the track is a rack by which the machine is fed along and constantly held up to the work. This feed is by means of an air-engine and worm-gearing on the truck. Another air-engine with another train of gearing slowly rotates the head. This head has a hub, a casing, four connecting-arms, and four bars which carry pneumatic hammers with chisel bits. Upon three of the bars six hammers are mounted, and upon the fourth bar there are seven, and as the head rotates they cut along different overlapping circles and thus cover the entire face of the excavation. Steel plates fixed between the groups of hammers are arranged to form pockets to catch and carry away the rock chips. These discharge into a hopper at the rear of the rotating head, and a conveyor leads to the rear of the machine. The designers estimate that this machine will be capable of removing 5,000 cu. ft. of rock per day, which would be equivalent to an advance of 100 ft., and, of course, three or four times as fast as by the drilling-and-blasting method. It is also calculated that $300 per day will be sufficient to meet all the expenses involved in operating the machine.

The Karns Machine.

This machine, the invention of J. P. Karns, formerly of Cripple Creek, may be called an enormous percussion-drill reciprocating a single cutter-head. The first machine tried in the Cripple Creek district used a column to support it. The company has at Magnolia, Colo., a 6-ft. machine mounted on a carriage. The head is 4 ft. in diameter, and is attached to a 9-in. shaft, which extends about 10 ft. to the back of the carriage, where it has a spherical direct connection with the 4-in. piston-rod of the actuating cylinder. The head rests on a cylindrical shield, which in turn bears on small wheels. The shield carries 3-in. balls on which the head rolls, and these balls rest on eighty $\frac{5}{8}$-in. balls. There is much more of complicated description which we cannot profitably follow. The cutters are made of bars 1 in. thick, 4 in. high, and of lengths varying from 9 to 24 in. Their faces are serrated so that there are a number of square pyramids on each. The cutters are arranged in three sets, the outer set having 24 cutters of medium length, the next set having 17, one being left out so that a man can get through the head, and the center set having 4 short cutters in advance of the others, which help to keep the machine in its path. An endless scraper, 27 ft. long, driven by a small three-cylinder engine, is provided to remove the cuttings, but it is expected that they can be flushed out by water.

This machine has actually done some work. A short run was made a year ago. The engine used was too large to follow in the bore of the 6-ft. cutter, and the compressors were too small to maintain more than one-half the pressure desired. The machine was run 6 min. 30 sec. for an advance of 2 in. The rock was a fairly-hard syenite, composed almost entirely of feldspar.

The Sigafoos Machine.

This machine, invented by R. B. Sigafoos, of Helena, Mont., is an electrically-driven machine, and the entire machine rotates upon wheels, which may be given a helical pitch to advance or withdraw the machine by its own rotation and to feed it slowly against the rock when in operation. The work is done by crushing or pulverizing the rock and washing it out with water under pressure. The rotating frame carries 10 crushing-heads, each mounted on a 4-in. shaft. These are

withdrawn by cam action, and a long, heavy helical spring shoots the shaft and crushing-head forward, the several heads striking in succession. The frame is made up of two heads connected by a 6-inch central rod and eight 1.5-in. tie-rods near the outside. The length over all is about 18 ft., and the weight is 20 tons; the weight is considered sufficient to hold it up to its work. Eight of the cutters are on the outer edge of the machine and two are near the center. The heads are about 2 ft. in diameter and about 5 in. thick. The teeth are not sharp and do not require sharpening. The cuttings are to be flushed out by a 3-in. stream of water under 40 lb. pressure, fed through the central shaft, and a spray-head keeps a $\frac{5}{8}$-in. stream behind each cutter. A 150-h.p. electric motor will be used, it being assumed that 65 h.p. will be required to start the machine, while half of that will be enough to keep it going.

The Fowler Machine.

The machine of George A. Fowler is one of the latest which has come to the knowledge of the public, and its scheme of operation is different from that of any of the others. It is expected to cut a rectangular face 7 ft. by 6 ft. 6 in., using 38 percussion-drills, all operating at once and all being moved about to cover the entire area while in rapid operation. The drills are arranged as a battery upon a massive cast-steel head hung by heavy hinges from the front of the machine. This head is to move back and forth to cover the entire face three times a minute, the drills striking at speeds up to 1,000 per minute. The movement of the head is not absolutely automatic, but can be controlled by the operator according to the requirements of the rock. There is no rotation of the bits, and the drills are not quite parallel, flaring enough at the outside to provide for necessary clearance and the maintenance of the full areas as the work progresses. The cutting-battery is framed in a shield which fits the periphery of the tunnel all around, inclosing all cuttings, which collect at the bottom, where they can be carried backward on to an elevating conveyor by the collective exhaust of all the drills and delivered to cars in the rear. The valve-motions of the drills and other details of this machine are novel and interesting, but we need not consider them here.

The Bennett Machine.

This machine, the invention of G. R. Bennett, of Denver, it is rather more difficult to take seriously than even some of the others. It carries 48 hammer-drills with 4-in. bits, drilling horizontally to a depth of 2 ft., then withdrawing and resetting laterally and again vertically until eight successive sets of holes have been drilled, thus covering a rectangular area of 8 ft. by 5 ft. 4 in. with 384 holes, all the shiftings and resettings being done automatically. The drills are set eight in a row horizontally with 8-in. centers, and they are spaced vertically with 4-in. centers in two sets of three each with 12 in. between the sets. The drive is electric, each vertical set or battery of six having one feed-cylinder and one driving-pinion, the sets striking alternately. The sets of drills are fed forward 2 ft.; as each set finishes its cut it is drawn back and the power is automatically shut off, and when all eight of the sets have thus finished their cuts the machine shifts over 4 in., and another cut is started. When this cut is done the drills are all dropped together 1 ft. and the previous drilling-operations are repeated. This finishes the upper half of the face; the drill-carriers now drop 3 ft., and a repetition of all the previous operations completes the entire face. It is thought that the webs of rock remaining between the holes will break down of themselves, but if they should remain until the series of holes are all drilled a man can go in and quickly break them down. It is not assumed that any of this breaking away could interfere with the constant automatic action of the drills and shifting-devices.

Development in the Size and Shape of Blast-Furnaces in the Lehigh Valley, as Shown by the Furnaces at the Glendon Iron Works.

BY FRANK FIRMSTONE, EASTON, PA.

(New Haven Meeting, February, 1909.)

IN the summer of 1842 my father, William Firmstone, was engaged by Charles Jackson, Jr., of Boston, to examine the conditions in the Lehigh valley as a site for blast-furnaces using anthracite for fuel. In consequence of his report, he was further engaged by Mr. Jackson to build a furnace for him and his partners on the Lehigh canal, 2 miles above the mouth of the river at Easton. Work was begun in the fall of 1842, and the first furnace blown-in in 1844. The history of the works, therefore, from 1844 to 1887, when my own connection with them ceased, covers only four years less than the whole period of the rise, culmination, and commencement of the decline in the smelting of iron with anthracite in America, the beginning of which, in a commercial sense, may be put in 1839.[1]

Although this paper, as the title indicates, is concerned almost exclusively with the furnaces at Glendon, still what was done there was more or less influenced by current opinion in the district, and even reflects, to some extent, the changes in opinion and practice with mineral fuel the world over.

No. 1 Furnace was built of red brick, on four piers, had three tuyeres and fore-hearth and tymp, as was universal with furnaces using mineral fuel until the introduction of Lürmann's cinder-tuyere. The dimensions and profile are shown in Figs. 1 and 2. The first hearth (Fig. 1) was of sandstone; all after were of fire-brick (Fig. 2). The profile was no doubt derived directly from Gibbons's furnace,[2] for it appears from W. Firmstone's note-books that in October, 1840, he visited "Gibbons' new furnaces building at Corbyn Hall," and I have an old drawing,

---

[1] *Trans.*, iii., 153 (1874–75).

[2] Percy, *Metallurgy of Iron and Steel*, p. 479 (1864).

made by him, marked "Corbyn Hall, 1841," practically the same as the figure above named in Percy. Although I have called this, with others, the Gibbons profile, essentially the same shape had been used in Sweden long before,[3] and something not very different in England at a still earlier date.[4]

Although what had already been done in this country, especially by David Thomas at the Crane works at Catasauqua, would have warranted building a much larger furnace, Mr. Jackson desired to make the first trial on a very modest scale. No. 1 worked well, and the building of No. 2 was begun in 1844 and the furnace blown-in in 1845. This furnace worked very well, used less fuel per ton of iron than No. 1, and naturally made much more iron. On the whole, it was for many years the best furnace at Glendon, especially in the very important point of regularity. The profile, Fig. 3, approximates the Gibbons shape, but the proportions are somewhat different from No. 1 and from Corbyn Hall. The dimensions were: Corbyn Hall, $H/d = 3.56$; No. 1, $H/d = 3.63$; No. 2, $H/d = 3.21$. No. 2 had four tuyeres. Excepting an increase in the diameter at the tuyeres from 4 ft. 9 in. to 5 ft., and a consequent slight steepening of the boshes, no important change was made in the profile of this furnace until 1869. Both No. 1 and No. 2 were blown by water-power, and the hot-blast ovens were fired with anthracite, as was the case at Catasauqua.[5] None of the gas was then utilized at Glendon.

The works were further increased in 1849 and 1850 by tearing down No. 1 and building a larger furnace with five tuyeres on the same site, and by building a third furnace, also having five tuyeres. By this time the use of the waste gases was well understood, and, accordingly, boilers and steam blowing-engines were erected to supply No. 3 Furnace, and to give more blast to Nos. 1 and 2, and all the hot-blast ovens were altered to be heated by the gases. Both boilers and hot-blast ovens were raised, some on cast-iron columns and some on masonry piers, to nearly the level of the flues, by means of which the portion of gas used was taken from the furnaces, very much as shown in

---

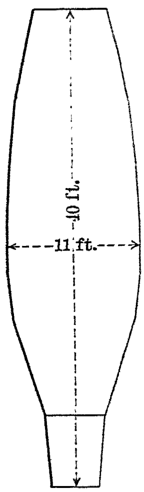
3 Braune, *Jern-Kontorets Annaler*, vol. lix., pp. 34, 35 (1904); and Jars, quoted in Beck's *Geschichte des Eisens*, part 3, pp. 355, 356.

4 Beck, *ibid.*, part 2, p. 970.

5 S. Thomas, *Trans.*, xxix., 908 (1899).

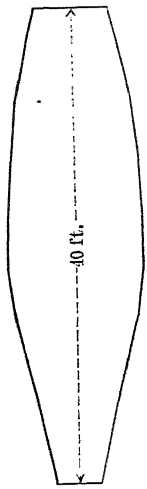
Mr. Thomas's paper, above cited, and which was, in fact, the standard construction for anthracite-furnaces in eastern Pennsylvania until the general introduction of closed tops in 1869 and 1870.

In the late '30's and early '40's there was, apparently, a strong tendency to increase the volume of the furnace, particularly of the upper part, and Gibbons had proved this to be correct



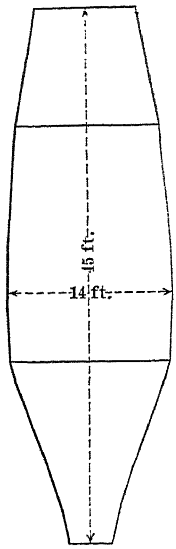
Blown-in, Mar. 15, 1844.
Blown-out, July 17, 1844.
Iron per week, tons, 52.30.
Average grade of iron, 1.68.
Fuel per ton pig, tons, 2.28.
Blast-temp., 500°–600°.
Blast-pressure, pounds, 4.
Number of tuyeres, 3.

FIG. 1.—FURNACE NO. 1.



Blown-in, Sept. 5, 1844.
Blown-out, May 16, 1846.
Iron per week, tons, 57.46.
Average grade of iron, 1.89.
Fuel per ton pig, tons, 2.04.
Blast-temp., 500°–600°.
Blast-pressure, pounds, 4.
Number of tuyeres, 3.

FIG. 2.—FURNACE NO. 1.



Blown-in, May 25, 1845.
Blown-out, Aug. 24, 1847.
Iron per week, tons, 68.36.
Average grade of iron, 3.27.
Fuel per ton pig, tons, 1.95.
Blast-temp., 500°–600°.
Blast-pressure, pounds, 4.
Number of tuyeres, 4.

FIG. 3.—FURNACE NO. 2.

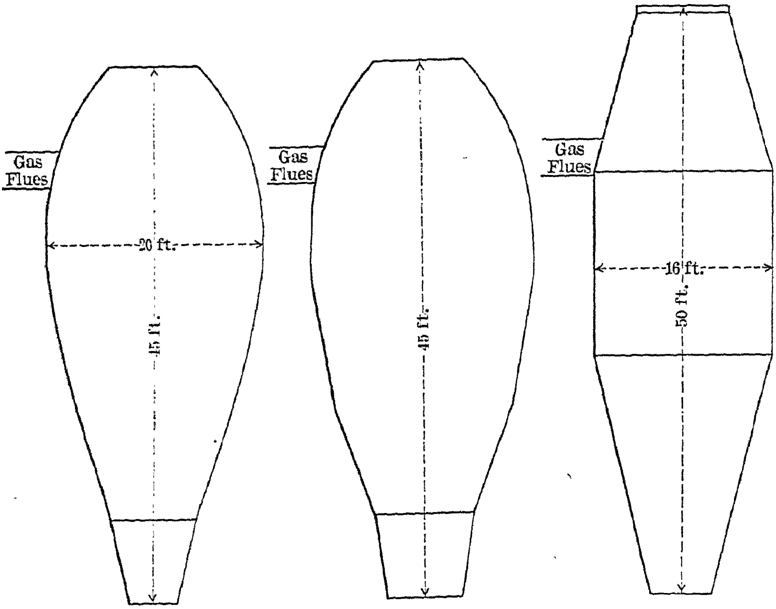
within limits by actual trial.[6] The same tendency was probably responsible for making furnaces cylindrical at the widest part for considerable heights, as, for instance, the furnace at the Crane works, shown in Mr. Thomas's paper.[7] In this connection, the following extract from a private note-book of W. Firmstone is of interest:

"Liverpool, Sept. 10, 1840, 3:30 p.m., went on change. Saw Mr. Jeavons. Said his furnace in South Wales had blown in with anthracite coal about a week

[6] Percy, *Metallurgy of Iron and Steel*, p. 477, *seq.* (1864).

[7] *Trans.*, xxix., 909 to 917 (1899).

since, doing well. Said they were a month after firing before they got the burden down to blow. Was surprised when I told him that in the States we blew a few hours after firing. Said his furnace was 46 ft. high, 12-ft. boshes; 8-ft. tunnel-head; cylindrical for 15 ft.—Dec. 10, 1840. (Carnbrae.) The furnaces are 8 ft. at top; four filling-places, 12 to 15 at boshes, being pretty straight to near the top, then cupped in all at once. This is the common plan in Scotland."



Blown-in, Mar. 2, 1851.
Blown-out, July 24, 1853.
Iron per week, tons, 114.04.
Average grade of iron, 3.42.
Fuel per ton pig, tons, 1.82.
Blast-temp., 500°-600°.
Blast-pressure, pounds, 4.
Number of tuyeres, 5.

Blown-in, Sept. 20, 1853.
Blown-out, Mar. 28, 1855.
Iron per week, tons, 140.69.
Average grade of iron, 4.14.
Fuel per ton pig, tons, 2.03.
Blast-temp., 500°-600°.
Blast-pressure, pounds, 4.
Number of tuyeres, 5.

Blown-in, July 1, 1855.
Blown-out, July 1, 1857.
Iron per week, tons, 131.47.
Average grade of iron, 2.72.
Fuel per ton pig, tons, 1.91.
Blast-temp., 500°-600°.
Blast-pressure, pounds, 4.
Number of tuyeres, 5.

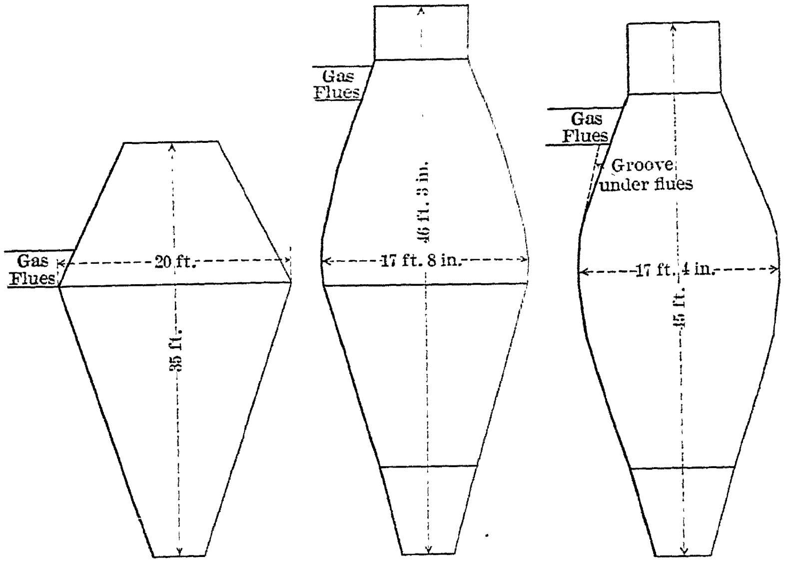
FIGS. 4, 5, AND 6.—FURNACE NO. 1.

This tendency was carried to the extreme in rebuilding No. 1, as shown by Figs. 4 and 5; but the results were not good, and after two blasts the furnace was raised from 45 ft. to 50 ft., and the diameter reduced to 16 ft., Fig. 6, with decided advantage.

There was also a notion at that time, in South Wales at least, that raw-coal- and anthracite-furnaces should be low and of relatively great diameter, no doubt with the expectation that the necessary blast-pressure would be less than with higher furnaces of the same cubic capacity. Such a furnace is shown

in Percy.[8] This notion was tested in building No. 3, Fig. 7, but the results were so decisively against it that it was soon blown-out and altered to the shape shown in Fig. 8.

The ill-effects which attend the dumping of the material in the center of an open-top furnace from a car with a trap-door bottom, or from a hopper closed by a small cone which is



Blown-in, Oct. 20, 1850.	Blown-in, June 10, 1851.	Blown-in, Aug. 26, 1862.
Blown-out, Mar. 31, 1851.	Blown-out, Aug. 7, 1853.	Blown-out, Apr. 21, 1865.
Iron per week, tons, 103.96.	Iron per week, tons, 109.12.	Iron per week, tons, 154.69.
Average grade of iron, 3.34.	Average grade of iron, 3.97.	Average grade of iron, 4.19.
Fuel per ton pig, tons, 2.30.	Fuel per ton pig, tons, 2.	Fuel per ton pig, tons, 1.82.
Blast-temp., 500°-600°.	Blast-temp., 500°-600°.	Blast-temp., 500°-600°.
Blast-pressure, pounds, 4.	Blast-pressure, pounds, 4.	Blast-pressure, pounds, 4.5.
Number of tuyeres, 5.	Number of tuyeres, 5.	Number of tuyeres, 5.

FIGS. 7, 8, AND 9.—FURNACE NO. 3.

raised to discharge the materials into a closed-top furnace, have often been noticed, and clearly traced to the rolling of the coarser part of the charge, especially the larger pieces of fuel, to the walls.[9] It is plain that filling with barrows into a furnace like Fig. 4 or Fig. 7 is not quite the same as dumping exactly in the center of the top, yet it is fairly evident that the great and sudden sidewise movement of the materials, in such cases, will result in an accumulation of the larger pieces at the

[8] *Ibid.*, Fig. 103, p. 562 (1864).

[9] De Vathaire, *Études sur les Hauts-Fourneaux*, p. 102 (1866); Chas. Cochrane, *Proceedings of the Institution of Mechanical Engineers*, pp. 163, 164 (1864).

walls, with the attendant bad effects. This unavoidable defect is one reason, probably the principal one, for the failure of such profiles. So far as I know, they are nowhere in use to-day. I have no doubt that for good work a furnace should widen downwards from the top at a moderate rate until about the middle of the total height is reached, but special reasons caused the adoption at that time ('50's and early '60's) in eastern Pennsylvania of shapes widening at what now seems an excessive rate. At all furnaces in the Lehigh valley and, so far as I know, elsewhere in the anthracite-regions, the gas for hot-blast ovens and boilers was then taken from the furnaces by a series of horizontal flues piercing the lining at depths of from 10 to 17 ft. below the open tops of the furnaces. By the rolling of pieces of the materials into the mouths of these flues, and by the accumulation of dust in the interstices of such pieces, the flues were greatly obstructed, and unless they were cleaned out at frequent intervals, a very troublesome operation, the flow of gas was greatly diminished or cut off completely. It is plain that this difficulty will be greatly diminished if the inner mouths of the flues come in a place where the furnace-walls batter inwards towards the top pretty sharply, and partly at least for this reason, many furnaces built at that time were thus drawn in at the top. Furnaces having this shape were not among those, so far as my own knowledge goes, which did notably good work in respect of fuel-consumption and regularity, and the application to them of the cup-and-cone, with no change in profile, no doubt aggravated the difficulties which at first, in most cases, attended the use of closed tops on the Lehigh.

A much better plan than the excessive contraction at the top was adopted at the Crane works, as early as 1850 according to information given me by S. Thomas shortly before his death. It was also used at the Thomas works, which were built in 1854–55. The furnaces widened from the top downwards at a moderate rate, but grooves, the width of the flues, were formed under each flue, so that a line following the natural slope of the materials, drawn from the top of the flue, did not cut the bottom, but came into the groove; thus no solid matter but the flue-dust could enter them, and they were easily cleaned. The excessive sidewise rolling of the materials was confined to

the space in front of each flue instead of occurring all around the circumference. This construction is clearly shown in the drawings of the Thomas works in Percy;[10] also in Wedding.[11] The same plan was used at the raw-coal furnace at Canal Dover, Ohio, by David Thomas, Jr., and was still employed at some of the raw-coal furnaces in the Shenango and Mahoning valleys in the late '60's and early '70's, no doubt derived from Canal Dover.

No. 3 Furnace was altered to this plan in 1862, but, as Fig. 9 shows, the top was drawn in excessively in order to get sufficiently deep grooves, and, in fact, although the furnace thus altered was an excellent gas-producer and greatly helped the other furnaces in this way, yet the consumption of fuel was always larger than in Nos. 1 and 2, in spite of a greater proportion of mottled- and white-iron in the product; the working also was decidedly less regular.

No important changes were made at Glendon from the '50's until 1867 and 1868, but at the Thomas works at Hokendauqua, which were built in 1854–55, the furnaces were made 60 ft. high. These furnaces are fully represented in the plates in Percy and Wedding above cited. Very full statistics concerning them were published by Prof. John A. Church in 1875.[12] The figures for fuel-consumption there given are not directly comparable with those here given for Glendon for like dates, because the furnaces at Glendon were burdened to make as much gray-forge as possible, while the object at Hokendauqua was to produce a very soft gray foundry-iron. Moreover, the ore-mixture at Glendon, consisting of more than half magnetic ores, was richer than that at Hokendauqua, which contained more than half of rather lean brown ores.

No. 5 Furnace at Glendon was designed and built in 1867-68. By that time the great saving in fuel which had resulted from increasing the cubic content, and especially from greater height in the furnaces, in the Cleveland district, had attracted general attention. The first design for No. 5 was for a furnace 60 ft. high and 18 ft. in diameter, but this was changed during construction to 72 ft. high and 18 ft. in diameter, the profile being

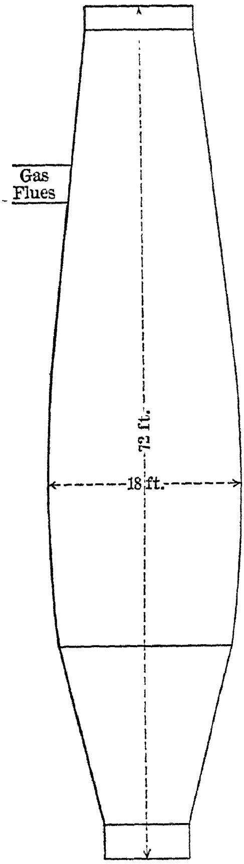
---

[10] *Ibid.*, p. 380, and the folding plates at end of volume (1864).

[11] *Ausführliches Handbuch der Eisenhüttenkunde*, 2d ed., vol. iii., p. 17 (1906).

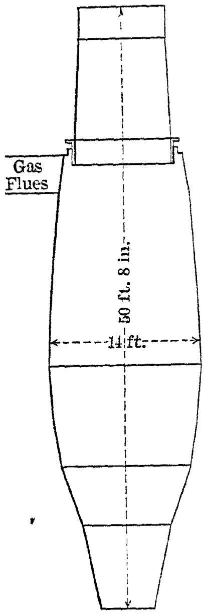
[12] *Trans.*, iv., 223 (1875–76).

as in Fig. 10. This profile differs but little from the Gibbons shape used in No. 1 in 1842, but H/d = 4, and the furnace belongs to Gruner's *élancé* type.



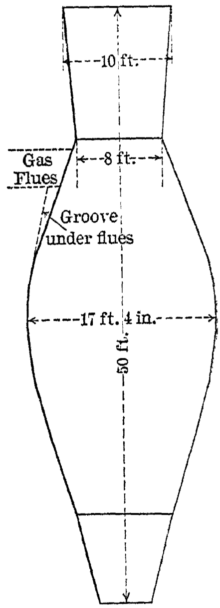
Blown-in, Jan. 7, 1869.
Blown-out, Feb. 28, 1871.
Iron per week, tons, 257.00.
Average grade of iron, 3.72.
Fuel per ton pig, tons, 1.35.
Blast-temp., 500°-650°.
Blast-pressure, pounds, 5.5-6.25.
Number of tuyeres, 7.

FIG. 10.—FURNACE NO. 5.



Blown-in, Nov., 1869.
Blown-out, Apr., 1873.
Iron per week, tons, 182.86.
Average grade of iron, 4.34.
Fuel per ton pig, tons, 1.59.
Blast-temp., 650°-750°.
Blast-pressure, pounds, 4.5-5.
Number of tuyeres, 4.

FIG. 11.—FURNACE NO. 2.



Blown-in, Aug. 22, 1869.
Blown-out, Feb. 6, 1871.
Iron per week, tons, 189.52.
Average grade of iron, 4.61.
Fuel per ton pig, tons, 1.48.
Blast-temp., 650°-750°.
Blast-pressure, pounds, 4.5-5.
Number of tuyeres, 5.

FIG. 12.—FURNACE NO. 3.

No. 5 Furnace was blown-in in January, 1869, and at once showed marked superiority over the older furnaces, both in fuel-consumption and in regularity. In consequence, Nos. 2 and 3 were raised to 50 ft. in height, with the profiles shown in Figs. 11 and 12. At the same time, larger and better hot-blast ovens

were built, and the combined effect of these changes was a decided but not revolutionary improvement in the results.

Fig. 13 is of No. 2 Furnace at the Musconetcong works, Stanhope, N. J., built in 1869–70, following the good work of

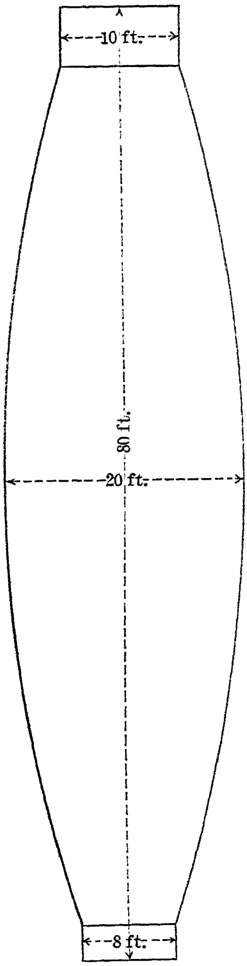
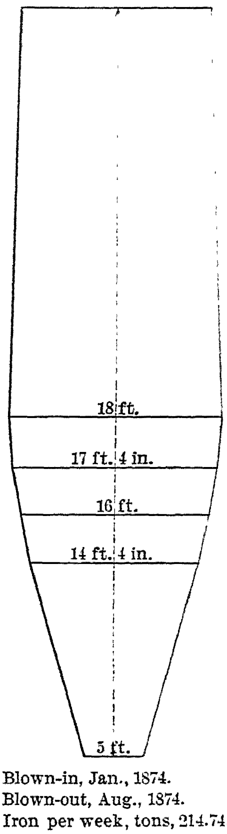


Fig. 13.—Furnace No. 2, Musconetcong Iron Works, Stanhope, N. J. 1871.



Blown-in, Jan., 1874.
Blown-out, Aug., 1874.
Iron per week, tons, 214.74.
Average grade of iron, 3.67.
Fuel per ton pig, tons, 1.52.
Blast-temp., 700°–800°.
Blast-pressure, pounds, 4.5–5.
Number of tuyeres, 5.

Fig. 14.—Furnace No. 1.

No. 5 Glendon, and laid out on similar lines and the same proportions ($H/d = 4$). This furnace always surpassed No. 1 at the same works, even after the latter had been raised from 55 ft. to 75 ft. in height. I cannot give the exact figures as to output and fuel-consumption, but the latter was very good,

although rather higher than for No. 5 Glendon; the ore-mixture was different and generally not quite so rich as at Glendon.

In 1871, Furnaces Nos. 2, 3, and 5 were blown-out because of the coal-strike of that year, and advantage was taken of the opportunity to change them from open top and side flue to the cup-and-cone plan of closed tops. The cup-and-cone had been applied to the furnaces of the Bay State Iron Co. at Port Henry, on Lake Champlain, in 1865 or 1866, according to T. F. Witherbee.[13]

I have elsewhere described[14] the very unsatisfactory result at first obtained, caused by our ignorance of the proper proportions for the bells.

The '70's of the past century were years of great importance for the study of the blast-furnace process, marked as they were by the publication of Bell's great work[15] and that of Gruner,[16] and the many comments on them. In both works, special emphasis was laid on the importance of the action of the gas at comparatively low temperatures in the upper part of the furnace, which, as previously stated, had been announced long before by Gibbons.[17] All this knowledge naturally revived the schemes for furnaces with a great volume in the upper part, by giving experimental support to what had been previously more or less vague conjecture, and as the capital importance of the mode of filling had just been emphasized by the troubles attending the first use of the cup-and-cone, it was natural to attribute the ill-success with the domed section at Furnace No. 1 in 1850 and 1852 to the nearly-central mode of filling into the relatively small top. Acting on these notions, Furnace No. 1 was raised, in 1873, from 50 ft. to 63 ft. in height, and lined up to a greatest diameter of 18 ft., with a top diameter of 16 ft., as shown in Fig. 14. The filling was effected through six small bell-and-hoppers disposed in a circle in a large ribbed plate covering the entire top of the furnace. The gas was taken off by a pipe in the center of this plate, which at the same time supported by brackets the six air-cylinders which moved the bells.

---

[13] *Trans.*, xxxviii., 894 (1908).

[14] *Trans.*, iv., 128 (1875–76); xiii., 520 (1884–85); and xxviii., 370 (1888).

[15] *Chemical Phenomena of Iron Smelting* (1872).

[16] *Études sur les Hauts-Fourneaux.*

[17] Percy, *ibid.*, p. 477 (1864).

This furnace was blown-in in January, 1874, and worked fairly well for some months, but showed no great improvement in fuel-consumption as compared with the work done before the alteration, and in this respect it was always far behind No. 5, which had practically the same cubic content, about 11,600 cu. ft. In July and August the furnace gradually worked less and less satisfactorily, and about August 9 it was almost completely scaffolded over, taking for some time only two or three charges per shift. Finally, it was blown-out when it appeared that, although nothing was left in it from the tuyeres to a height of about 15 ft. above them, at that level it was bridged over, the lower surface of the scaffold being nearly flat and level. The only opening through the scaffold, which was about 10 ft. thick, was a somewhat-tortuous hole 5 or 6 ft. in diameter next the walls on the right-hand side when facing the tymp. In trying to remove the scaffold quickly, water was so freely and incautiously used that the slaking of the lime in the suspended mass, by its expansion, completely wrecked the red-brick masonry, and no course remained but to tear the furnace down to the foundation.

This furnace departed so much from ordinary lines, both in shape and in method of filling, that it is practically useless to speculate on the reasons for its failure. Possibly, probably even, a much greater diameter at the tuyeres would have helped matters. Had we succeeded in clearing out the scaffold without injury to the furnace, as we could have done with more patience and discretion in the use of water, it would have been proper and desirable to make another trial, without altering the shape or the method of filling; but during the short blast nothing had been observed to warrant the building of a new furnace on plans so different from those known to give at least good results. Accordingly, in rebuilding, the shape and dimensions shown in Fig. 15 were adopted, for it was not possible to build a duplicate of No. 5 without enlarging the foundations and tearing down adjoining buildings, which it was very desirable not to disturb. The red-brick work of the furnace was finished in November, 1875, but the lining was not put in until 1877. The furnace was blown-in in August, 1877, and gave excellent results, working regularly and with very low fuel-consumption, but, on the whole, no better than

No. 5, and perhaps not quite so well. Unluckily, direct comparison over a considerable period is not possible, because, during and after 1877, No. 5 ran much of the time on foundry-iron, and on a different ore-mixture from that used in No. 1. It never happened, however, when both were running at the same time on forge-iron, and therefore on identical ore-mixtures, that No. 5 did not do somewhat better as regards fuel-consumption than No. 1, and naturally, being considerably larger, No. 5 made rather more iron per week. John Fritz, in the extension of the Bethlehem Iron Works between 1870 and 1875, made the new furnaces 70 ft. high and 16 ft. in greatest diameter. These furnaces worked well in spite of a very variable iron-ore supply.

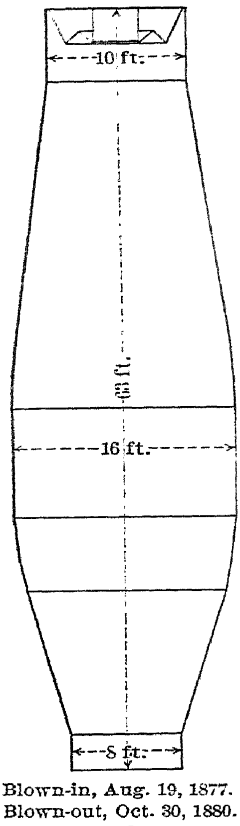
In 1879 it was decided to replace Nos. 2 and 3 by larger and higher furnaces, the advantages of which had been incontestibly shown by ten years' experience with No. 5. Drawings for this change had been prepared under my father's direction in 1876, which resulted, under the influence of Gruner's work and the unfavorable outcome at No. 1 in 1875, in a design for a furnace 80 ft. high and 18 ft. in greatest diameter. It differed from No. 5 in having the cylindrical part at place of greatest diameter, 6 ft. high instead of 3 ft., and the cylinder at the top 7 ft. high instead of 2 ft., as in No. 5. This design belongs to Gruner's *élancé* type, H/d being even greater than 4.

It is a good plan to have a cylinder of considerable height at the top, because it decreases the irregularities in distribution which arise when the furnace is not kept very exactly full to the proper stock-line.[18]

In 1879, in actually rebuilding the furnaces, beginning with No. 3, we had to consider, in addition to the experience previous to 1875, the excellent results during 1877–78 with the rebuilt Furnace No. 1, which had a top 10 ft. in diameter for a greatest diameter of 16 ft. It seemed probable that this form contributed to the good working, and that it would be well, therefore, to adopt a top diameter of 11 ft. for the new furnace, which is practically in the same proportion to 18 ft. as 10 ft. is to 16 feet.

The final design was for a furnace 81 ft. high, 18 ft. in greatest diameter, and 11 ft. in diameter at top, with the Coingt

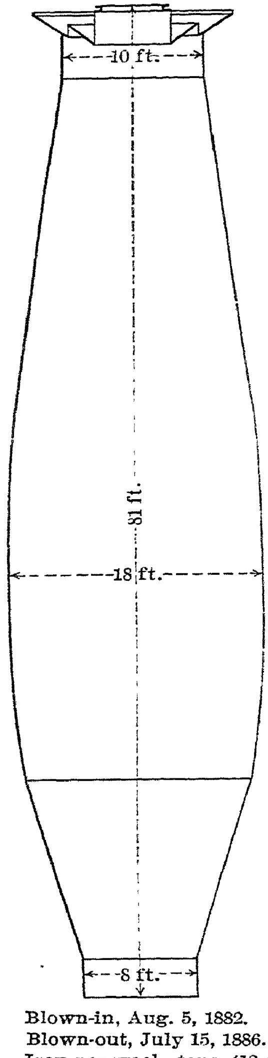
[18] Gjers, *Journal of the Iron and Steel Institute*, vol. ii., p. 211 (No. 4, 1871).



Blown-in, Aug. 19, 1877.
Blown-out, Oct. 30, 1880.
Iron per week, tons, 304.2.
Average grade of iron, 3.13.
Fuel per ton pig, tons, 1.15.
Blast-temp., 800°–850°.
Blast-pressure, pounds, 6.5–7.5.
Number of tuyeres, 5.

Same profile, Langen charger:
Blown-in, Mar., 1881.
Blown-out, Apr., 1884.
Iron per week, tons, 340.2.
Average grade of iron, 3.01.
Fuel per ton pig, tons, 1.21.

FIG. 15.—FURNACE No. 1, REBUILT 1875–77.



Blown-in, Aug. 5, 1882.
Blown-out, July 15, 1886.
Iron per week, tons, 412.4.
Average grade of iron, 2.75.
Fuel per ton pig, tons, 1.21. (one-eighth coke.)
Blast-temp., 800°–850°.
Blast-pressure, pounds, 8–9.
Number of tuyeres, 7.

86 weeks unsized ore (1882–84):
Iron per week, tons, 349.5.
Average grade of iron, 2.53.
Fuel per ton pig, tons, 1.32.
118 weeks sized ore (1884–86):
Iron per week, tons, 458.3.
Average grade of iron, 2.88.
Fuel per ton pig, tons, 1.15.

FIG. 16.—FURNACE No. 2, REBUILT 1881–82.

charger.[19] This furnace was blown-in in October, 1880. No illustration of it is given, since it differs from the new No. 2, Fig. 16, only in the diameter at the top, and the use of a Coingt instead of the Langen charger. In fact, No. 3 was altered to 10 ft. on top and the Langen charger substituted for the Coingt in May, 1882. The change in shape at the top was effected without blowing-out, by hanging a cast-iron liner, of proper shape, from the lining-ring. The change appreciably improved the working of the furnace.

By the time we were ready to rebuild Furnace No. 2, we had proved the advantages of the modified Langen charger,[20] and it was, of course, adopted for the rebuilt furnace. No. 2 was blown-in in July, 1882.

Neither of the rebuilt furnaces gave really satisfactory results, both showing decidedly greater tendency to scaffold and to work irregularly than either No. 1 or No. 5, more especially No. 3, before the change to 10-ft. top and the Langen charger. Moreover, the necessary blast-pressure was decidedly higher, from 9 to 11 lb. per sq. in., as compared with from 6 to 8 lb. for Nos. 1 and 5. This latter disadvantage was greatly diminished by the use of one-eighth of coke, which we began soon after the blowing-in of No. 2. A small gain in regularity and an increased output also followed this change.

The new No. 3 was run for a considerable part of the time on foundry-iron, hence the figures for fuel-consumption and output are not properly comparable with those for Nos. 1 and 2, and are therefore not given.

It was certain that the inferior results at Nos. 2 and 3 were not due directly to the increased height, and any consequent increase in the crushing-strain on the materials, a matter about which much has been said and written, but, in fact, very little in the way of exact observation has been published, because, first, the actual height of the column of materials in the new furnaces, allowing for the height occupied by the charging-apparatus, was not sensibly greater than it had been in No. 5 during the time it was worked with open top; and second, No. 2 at the Musconetcong works, Fig. 13, always did better than No. 1 at the same works, No. 1 being 75 ft. high

[19] *Trans.*, xxviii., 370, Figs. 3, 4, and 5 (1898).

[20] *Trans.*, xiii., 520 (1884–85).

and No. 2, 80 ft. Moreover, there was a falling-off in the work of all the furnaces at Glendon, after the middle of 1879, which pointed pretty clearly to something which affected all of them in the same manner but in a different degree. The cause was finally discovered in a serious increase in the proportion of very fine magnetic ore from the mines in New Jersey, due to the incautious use of dynamite, and to great irregularity in the percentage of fine and coarse at different times, depending on whether at the time of shipment the stock of ore under the trestles at the mines was increasing or diminishing. By limiting the use of dynamite to narrow work in sinking and driving, which, by the way, did not sensibly increase the cost of the ore, the proportion of fine ore was much reduced, and by screening over $\frac{7}{8}$-in. bar-screens and shipping the fine and coarse in separate cars, it was possible, within limits, to control the percentage of fine and coarse filled into the furnaces.

At the furnaces a charge was made up of fine or of coarse ore; the two sizes were never mixed in the same round. This was done, at first, because it was much easier to be sure that a certain number of charges of fine ore were filled per shift than that so many barrows in each and every charge during a shift were of fine ore; moreover, it insured that the fine was uniformly distributed around the circumference of the furnace, and not, perhaps, filled all on one side. The resulting improvement was surprising, as is shown from the data given in Fig. 16. It was far greater than could possibly arise merely from the decrease in the percentage of fine material. T. F. Witherbee noted long ago the desirability of regularity in the proportion of fine to coarse ore;[21] E. L. Uehling has strongly insisted on the advantages of sizing and charging each size by itself, and not mixed with another in the same round; and E. Belani has done the same.[22] I can emphatically confirm, from my own observation, all that these authorities say as to the resulting advantages.

The use of sized ore in the other furnaces wrought appreciable improvement, but it was not so remarkable as in Nos. 2 and 3.

In spite of the good result shown by No. 2, following the

[21] *Trans.*, iv., 377 (1875–76).

[22] *Stahl und Eisen*, vol. xxiii., No. 13, p. 777 (July 1, 1903).

use of a little coke, and especially after sizing the ore, the new furnaces must be regarded as decidedly inferior to Nos. 1 and 5, particularly in regularity. The probable cause of this inferiority I do not pretend to indicate, but, considering the good results from No. 2 at Musconetcong, it may be in the too-great ratio of the diameter to the height, H/d being 4.5 in the Glendon furnace, as compared with $H/d = 4$ at Musconetcong.

The matter seems to show very well the possible cumulative effect of comparatively small errors; that is, in this case the ill-effect of an increase in the fine ore used was not very marked in Nos. 1 and 5, but was serious in Nos. 2 and 3, when to it was added the effect of a change for the worse in the profile.

In comparing the fuel-consumption and the average grade of the iron made after 1875 with that before that year, it is necessary to remember the revolution in the production of finished materials, especially of rails, which followed the development of the Bessemer process. Before 1875 there was always a great market for white iron for the rail-mills, and it was often made intentionally; after that it was commonly necessary, as far as possible, to avoid producing it, and, in general, the iron used for puddling was much grayer than that previously employed. This, of course, tended to increase the average fuel-consumption at the furnaces and to decrease the output.

As regards the increased fuel-consumption at No. 1, after 1880, this was probably due, in part, to the very considerable increase in the weekly output, as compared with that from 1877 to 1879.

The Laws of Fissures.

BY BLAMEY STEVENS, SEATTLE, WASH.

(New Haven Meeting, February, 1909.)

THE object of this paper is to present a theory of the formation of fissures, which seems to be supported by all available data. The investigation is, in the main, an exact one, and irregularities of the rock-structure are generalized. From the conclusions drawn a general classification of fissures is made, and this should be of practical use in the comparison of mineral deposits. The theory also throws some light on the equilibrium of the earth's crust and the depth of action of surface-waters. Incidentally, additions are made to the theory of earthquake-slips.

Former Theory.

It has been previously pointed out that the "jointing" of homogeneous rocks occurs in two sets of planes, at right angles to one another, and making angles of 45° with the directions of greatest and least principal stress.[1]

The theory by which this fact is explained does not, however, account for fissures and faults, because normal faults are usually more vertically, and reverse faults more horizontally, inclined than 45°.

The "jointing law" is based on the following axiom: "In a homogeneous mass under pressure, rupture must take place on the lines of maximum tangential stress."[2] This law applies, with fair approximation, to the tests on jointing which can be made in the laboratory, and as nearly as can be expected to the natural jointing-planes in rock-masses, where the stresses are not very great; but it entirely fails to account for fissure-fractures.[3]

---

[1] G. F. Becker in *Bulletin of the Geological Society of America*, vol. iv., p. 48 (1892). See also, Torsional Theory of Joints, *Trans.*, xxiv., 130 (1894).

[2] *Ibid.*

[3] Rankine, *Applied Mechanics*, art. 283, sec. ii.

Author's Revised Theory.

It is generally assumed that these joint-planes are fissures and faults in embryo, but I shall show in this paper that an entirely different law governs the formation of faults and fissures. This law may be conveniently known as the fissure law, and may be stated as follows:

In a homogeneous mass under pressure, slipping tends to take place only along those planes on which the ratio of tangential stress to direct stress is equal to the coefficient of friction of the material sliding on itself.

The law is proved by observations on actual rock-masses and by laboratory-experiment.

Exact Analysis.

The condition of stress in any one place in a rock-mass or other solid substance may be completely represented by three principal stresses (p_x, p_y, p_z) at right angles with one another, illustrated in Fig. 1.

Let us assume $p_x > p_y$ and $p_y > p_z$; and in Fig. 1 let the plane of the paper be at right angles to the medium principal stress, p_y. Now let us consider the action of forces and stresses on a plane, GH, which is also supposed to be perpendicular to the plane of the paper. The stresses on any plane must be clearly distinguished from the forces on it, the stress being the amount and direction of force per unit-area of plane. If p_x and p_z represent principal stresses (as indicated by the equally-spaced arrows), the total force parallel to p_x on the plane GH is GH $\times$ y $\times$ $p_x \sin\theta$, where y is the breadth of the plane GH, measured parallel to the y axis. This may be resolved into GH $\times$ y $\times$ $p_x \sin^2\theta$ normal to the plane GH, and GH $\times$ y $\times$ $p_x \sin\theta\cos\theta$ tangential to it. Similarly, the total force parallel to p_z on the plane GH is GH $\times$ y $\times$ $p_z \cos\theta$, and this may be resolved into GH $\times$ y $\times$ $p_z \cos^2\theta$ normal and GH $\times$ y $\times$ $p_z \cos\theta \sin\theta$ tangential. Hence, the total normal force is GH $\times$ y $(p_x \sin^2\theta + p_z\cos^2\theta)$, and the total tangential force is GH $\times$ y $(p_x \sin\theta\cos\theta - p_z\cos\theta\sin\theta)$. As all these forces are distributed over an area GH $\times$ y, the total normal stress is

$$\left.\begin{array}{l} p_x \sin^2\theta + p_z \cos^2\theta \\ \text{or } \dfrac{p_x + p_z}{2} - \dfrac{p_x - p_z}{2}\cos 2\theta, \end{array}\right\} \qquad (1)$$

and the total tangential stress is

$$\left.\begin{array}{l} (p_x - p_z)\sin\theta\cos\theta \\ \text{or } \dfrac{p_x - p_z}{2}\sin 2\theta. \end{array}\right\} \qquad (2)$$

It may be easily seen that expression (2) forms a maximum when $\cos 2\theta = 0$, *i.e.*, when $\theta = \pm 45°$. These are the rectangular joint-planes formed when the tangential stress (p_t), necessary to rupture, is reached, *i.e.*, when

$$\frac{p_x - p_z}{2} \sin 2\theta = p_t. \qquad (3)$$

Or, putting $\theta = \pm 45°$, we get $\pm \frac{p - p_z}{2} = p_t$ At the same time the normal stress is $\frac{p_x - p_z}{2}$.

Now if a coefficient of friction be assumed at unity, slipping can only begin when the tangential and normal stresses are equal, *i.e.*, when $\pm (p_x - p_z) = p_x + p_z$. This is only possible when $p_x = 0$ or ∞, or $p = 0$ or ∞, neither of which is a possible condition; that is to say, with a coefficient of unity joint-planes cannot possibly form faults.

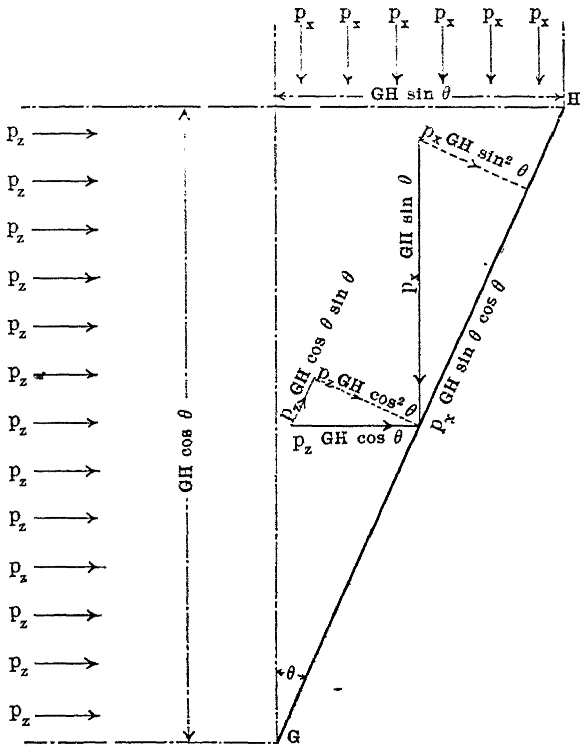


FIG 1.—DIAGRAM SHOWING RESOLUTION OF PRINCIPAL STRESSES ON AN OBLIQUE PLANE.

Let us now proceed to find the inclination of the fracture which is most favorable to the formation of fissures or faults. In order that slipping may occur between the surfaces of any fissure, the rules of frictional stability must be obeyed; that is to say, the coefficient of friction multiplied by the normal component of stress must be equal to the tangential stress.

In symbols, $a\,(p_x \sin^2 \theta + p_z \cos^2 \theta) = (p_x - p_z) \sin \theta \cos \theta \qquad (4)$
where a is the coefficient of friction.

It is evident that there is a ratio between p_x and p_z which will vary as θ and a are varied; this ratio is

$$\frac{p_x}{p_z} = \frac{\sin \theta \cos \theta + a \cos^2 \theta}{\sin \theta \cos \theta - a \sin^2 \theta} = \frac{1 + a \cot \theta}{1 - a \tan \theta} \quad (5)$$

One of the principal stresses is supposed to be vertical; its mean value must, on the average, be dependent on and equal to the weight of overlying material; and under a certain range of conditions more fully dealt with elsewhere in this paper, one of the principal stresses, which is horizontal, will tend to be as great or as small as is consistent with occasional slipping along fissures when the limiting ratio of principal stresses is reached. This limiting ratio depends on a and θ, and has a maximum with respect to θ which will be found to be given by the equation $\cot 2\theta = a$. If a be fixed at $= 0.9$, which seems to be the mean value most in conformity with observed data, then

$$\theta = \pm 24°.$$

Result of Analysis.

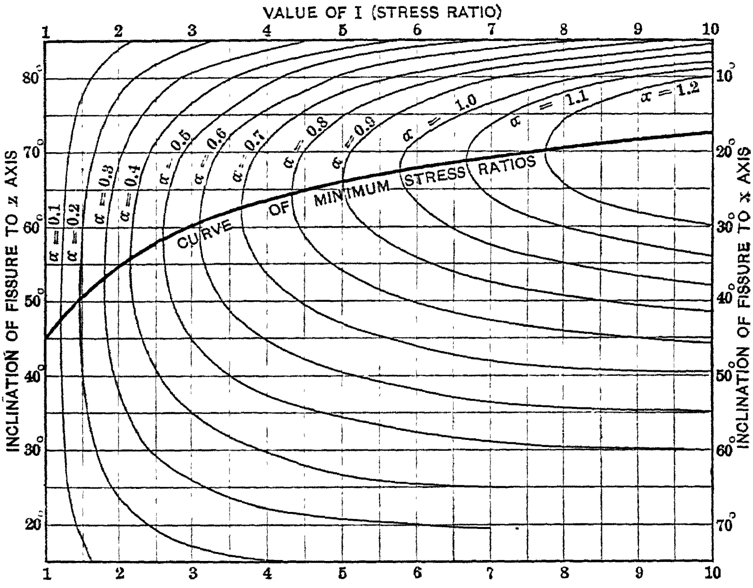
What I have demonstrated in the foregoing mathematical argument is that, if there be fractures at all inclinations in a rock-mass, then, with a coefficient of friction of $\alpha = 0.9$, slipping will occur only along those inclined at 24° to the axis of greatest principal stress. It will be seen from the diagram, Fig. 1, that if the axis of greatest principal stress is vertical, the displacement along the fissure will be that of a normal fault, and the dip of the normal fault which is most favorable to slipping will be 90 — 24 = 66°. Similarly, when the axis of greatest principal stress is horizontal, the displacement along the fissure is that of a reverse fault, and the dip most favorable to slipping is 24°.

TABLE I.—*Ratios of Greatest to Least Principal Stresses Necessary to Produce Slipping in Fissures.*

Inclination to Least Principal Stress (90—θ).	Coefficient of Friction (Statical). 0.1	0.2	0.3	0.4	0.5	0.6	0.7	0.8	0.9	1.0	1.1	1.2	Inclination to Greatest Principal Stress (θ).
90°	∞	∞	∞	∞	∞	∞	∞	∞	∞	∞	∞	∞	0°
85°	2.16	3.32	4.54	5.78	7.01	8.30	9.56	11.0	12.2	13.6	15.0	16.4	5°
80°	1.60	2.21	2.85	3 51	4.20	4.91	5.67	6.44	7.09	8.20	9.00	9.89	10°
75°	1.41	1.85	2.31	2.80	3.42	3 87	4.43	5.07	5.76	6.47	7.24	8.10	15°
70°	1.32	1.67	2.04	2.45	2.91	3.38	3.92	4.52	5.15	5.89	6.72	7.65	20°
65°	1.27	1.58	1.91	2.27	2.71	3.18	3.71	4.34	5.03	5.88	6.89	8.10	25°
60°	1.24	1.52	1.84	2.21	2 63	3.12	3.70	4.45	5.33	6.48	7.96	10.0	30°
55°	1.23	1.50	1.81	2.18	2.64	3.19	3.92	4.87	6.18	8.09	11.2	23.2	35°
50°	1.22	1.48	1.82	2.22	2.75	3.46	4.46	5.96	8.45	13.6	30.0	∞	40°
45°	1.22	1.50	1.86	2.33	3.00	4.00	5.67	9.00	19.0	∞	∞		45°
40°	1.24	1.54	1.95	2 55	3.51	5.27	35.4	∞	∞				50°
35°	1.25	1.60	2.12	3.00	4.74	9.90	∞						55°
30°	1.28	1.71	2.45	4.00	9.61	∞							60°
25°	1.33	1.92	3.20	8.37	∞								65°
20°	1.43	2.39	6.37	∞									70°
15°	1.64	4.15	∞										75°

This table is constructed from formula (5). The following example illustrates its use:

It is wished to find the horizontal stress which has caused slipping in a normal fault, dipping at 75°, the coefficient of friction being estimated at 0.7 and the mean depth at 10,000 ft. The greatest principal stress will be vertical and the least principal stress horizontal, so that the dip will be the same as the inclination to the least principal stress. Opposite 75° in the first column and under the coefficient of friction (0.7) we find the figure 4.43. The least horizontal stress is therefore equivalent to a depth of rock of $\frac{10,000}{4.43} = 2,257$ feet.



This diagram corresponds with Tables I. and II. The curve of minimum stress ratios gives the most favorable angle of slipping for any given coefficient of friction.

FIG. 2.—DIAGRAM SHOWING ISO-FRICTION CURVES AND CURVE OF MINIMUM STRESS RATIOS.

Table I. and the corresponding diagram, Fig. 2, show the dips of fissures consistent with any coefficient of friction (α) and ratios of stress $\left(\frac{p_x}{p_z} \text{ or I.}\right)$

A larger ratio of the principal stresses will cause slipping, but for equilibrium the ratio I must be smaller than the one indicated.

It will be seen from the diagram that in general there are two angles at which slipping will just occur; but for a minimum stress-ratio these two become coincident, and the corresponding angle is then known as "the most favorable angle of slip" for the particular coefficient of friction.

If the stress-ratio is reduced below the minimum no slipping can occur at any angle with the given coefficient of friction. Conversely, as the stress-ratio of a rock-mass fractured at all inclinations is increased from a low equilibrium value, slipping takes place at "the most favorable angle of slip."

A fissure well lubricated by talc would have a smaller coefficient of friction than 0.9; in this case the most favorable dip of the fissure might be less than 66° or more than 24°.

Complete Problem.—Let us now try to find how the stresses in a solid rock-mass could break a new fissure along a plane which would be favorable to combined breaking and slipping. We will presume that the tangential stress on the plane along which fracture is occurring is just sufficient to overcome the sum of friction and fracture. Referring to expressions (1) and (2), we therefore construct the following equation:

$$\frac{p_x - p_z}{2} \sin 2\theta = a \left(\frac{p_x + p_z}{2} - \frac{p_x - p_z}{2} \cos 2\theta \right) + p_t \qquad (6)$$

The ratio $\frac{p_x}{p_y}$ then becomes a maximum when $\cot 2\theta = a$. This is exactly the same result as when simple sliding without fracture was considered (see Table II.).

Let us now examine the theory in the light of experimental tests made in the laboratory by the application of simple crushing-tests. In these cases $p_z = 0$, but by our assumption (from equation 6)

$$p_z = \frac{p_x (\sin 2\theta + a \cos 2\theta - a) - 2\, p_t}{\sin 2\theta + a \cos 2\theta + a}. \qquad (7)$$

or putting $\theta = 24°$ and $a = 0.9$

$$p_z = \frac{0.2227\, p_x - p_t}{1.1227}. \qquad (8)$$

If p_x is measured by the depth, or head of rock causing the pressure, then p_t, in equivalent units, will be 1,600 ft. for granite.

In Fig. 3, the strong lines are constructed from equation (8) with $p_t = 1{,}600$ ft., whence it will be seen that when $p_z = 0$ then $p_x = 7{,}200$ ft., which corresponds very closely to compression rupture as determined by testing-machine.

Similarly, the dotted lines in the diagram are constructed from equation (3), viz.: $\frac{p_x - p_z}{2} = p_t$, whence, putting $p_z = 0$, we get $p_x = 3{,}200$ ft. This figure also corresponds very closely with the "cracking" noted by observers in the same laboratory crushing-test.

The above investigation shows how the fracture of fissures may take place, and that this theory of fissures is supported by the results of laboratory compression-tests of rock-specimens, if $\alpha = 0.9$. If θ is the inclination of the fracture to the greatest principal stress, then $\cot 2\theta = \alpha$, gives the value of θ which is the most favorable to slipping, and from experimental tests this seems to correspond very closely with the value of θ which is most favorable to fracture.

Table II. and the heavy line in Fig. 2 show the value of the stress-ratio, I, for all values of θ or α, for either fracture or the most favorable slipping-conditions.

TABLE II.—*Inclinations and Stress-Ratio Constants for Combined Fracture and Slipping of Fissures.*

Coefficient of Friction. α.	Inclination to Axis of x. θ.	Inclination to Axis of z. $90-\theta$.	Constant I = $\frac{p_x}{p_z}$ when $p_t = 0$.	Constant B = $\frac{p_x}{p_t}$ when $p_z = 0$.
0.0	45° 00′	45° 00′	1.00	2.0
0.1	42° 09′	47° 51′	1.22	2.2
0.2	39° 21′	50° 39′	1.48	2.4
0.3	36° 39′	53° 21′	1.81	2.7
0.4	34° 06′	55° 54′	2.18	3.0
0.5	31° 43′	58° 17′	2.62	3.2
0.6	29° 31′	60° 29′	3.12	3.5
0.7	27° 30′	62° 30′	3.68	3.8
0.8	25° 40′	64° 20′	4.34	4.1
0.9	24° 00′	66° 00′	5.00	4.5
1.0	22° 30′	67° 30′	5.8	4.8
1.1	21° 08′	68° 52′	6.7	5.2
1.2	19° 54′	70° 06′	7.6	5.5

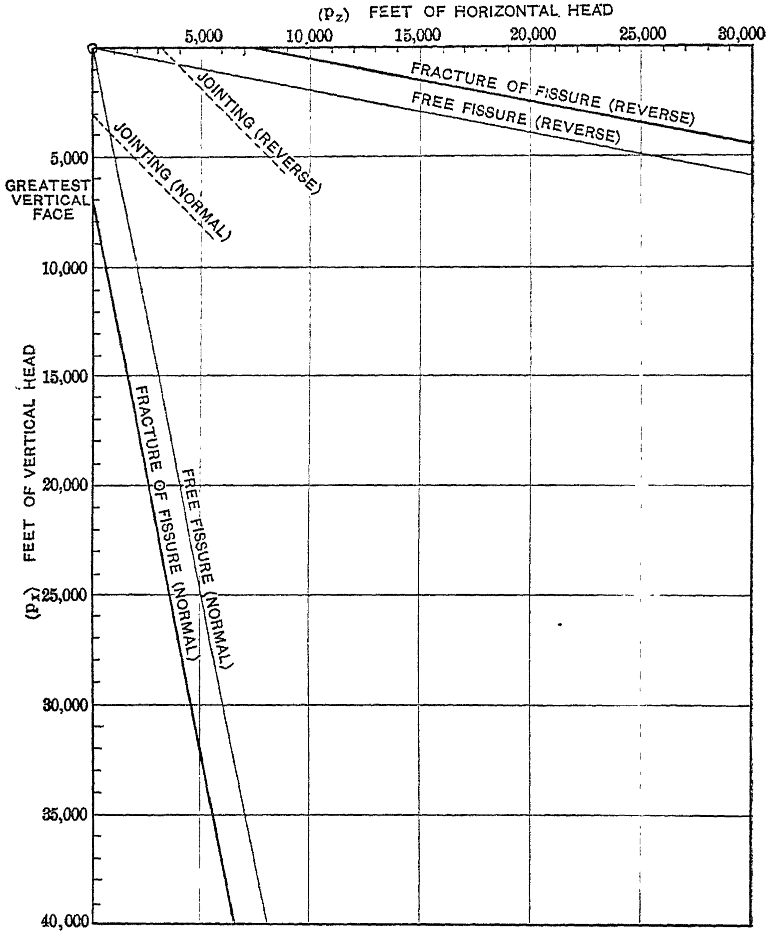
NOTE.—The formula for fracture making use of above constants is

$$p_x = Ip_z + Bp_t.$$

Fig. 3 shows the greatest horizontal stress compatible with fracture when the coefficient of friction is 0.9 and the greatest principal stress is vertical. The theoretical dip of the fracture is 66°, the same dip as that previously determined as the most favorable to slipping between the walls of a normal fault. The dotted line shows the greatest horizontal stress compatible with cracking or the formation of joint-planes dipping at 45°. The diagram illustrates also the comparative unimportance of fracture, as compared with friction, by the proximity to one another of the lines of fracture and free fissures.

Limitations of Jointing.

The phenomenon of jointing belongs essentially to brittle matter; in tough substances there is either no jointing at all or it manifests itself at angles of less than 45° with the greatest principal stress. The inference is, that friction interferes with



Constants: $p_t = 1{,}600$ ft. head; $\alpha = 0.9$.

FIG. 3.—VERTICAL AND HORIZONTAL STRESS DIAGRAM FOR AN AVERAGE GRANITE.

fracture before fracture actually occurs, as we have assumed in equation (6), which we should expect to be nearer the truth for tough rocks than for brittle ones. It would also seem that rock-stresses may become too great for jointing to be formed.

Elimination of Cohesion.

The laws of sand-equilibrium as deduced by Rankine[4] are a special case of the rock-laws as discussed in this paper, and are obtained from them by equating to zero the tangential stress necessary to fracture the rock. It will be noticed that this condition is also approached when the actual rock-stresses are very large; so that at considerable depths this tangential resistance may be disregarded, and we may illustrate the rock as acting like so much sand, held together only by gravity and friction. Considering the smallness of the simple tangential stress necessary to fracture rock in comparison with the evident great thickness of the earth's crust, we must therefore conclude that the earth as a whole owes no noticeable amount of its stability to cohesive forces. In this connection we find that Lord Kelvin, in his calculation that the earth must be more rigid than steel to resist the distortion due to the tidal forces of the sun and moon, uses the term "rigid" only in the sense that sand is rigid under gravitational forces and friction, and he does not presume any cohesion to exist.[5]

Varieties of Fissures.

Let us now consider what differences there may be in the character of a fracture near the surface and at great depth. Some amount of energy must be necessary to fracture. Near the surface, this energy will be considerably in excess of that required to start a slip on a fissure previously existing. The fractured surfaces will therefore tend towards a minimum of area; consequently, they will be clean cut and not multiple. On the other hand, at great depths, where the cohesional forces are of no relative importance, the rock will be always ready to form new fissure-courses, and each of these may be a complex and multiple fracture. Intermediate between these limiting forms of fissures we have every gradation. Various terms have been employed for these features, but the following three are perhaps sufficiently distinctive.

Varieties of Fissures.	True or clean-cut fissures, formed at small depths.
	Fissures with false walls, formed at medium depths.
	Shear-zones, formed at great depths.

[4] *Applied Mechanics*, art. 194 *et seq.*

[5] Geikie's *Text-Book of Geology*, book ii., part i.

The plastic metamorphism evidenced in the structure of many rocks may also be elucidated in part by the comparison with sand, but other mechanical, physical, and chemical forces undoubtedly have their influence on the change effected.

It is evident that the depth at which the above varieties of fissures are found in a homogeneous formation is proportional to the tangential stress necessary to fracture, expressed in linear measurement head of rock; thus, in sandstone a certain variety of fissure would be formed at from a half to a quarter of the depth at which the same variety would be formed in granite.

Classes of Fissures.

Let us now find how the ratio of vertical to horizontal thrust in the rock-mass adjusts itself. There are certain external conditions to consider. Chief among these is gravity, the action of which justifies us in assuming that over a moderate area of country, we may depend upon the mean vertical component of stress, at any level, being a direct stress, and represented by the depth or head of rock, H, above the level considered.

The horizontal stresses will be either greater or less than the vertical stress or depth, H. If less, the least of them must not be more than $\frac{H}{I}$ (where I is the ratio shown in Table II.) for any adjustment by slipping to take place. The solvent action of the surface-waters circulating through the cracks and crannies of the rock-mass tends continually to shrink the whole mass. This relieves the horizontal stress until it cannot balance the vertical stress H, because it has fallen slightly below $\frac{H}{I}$. Slipping then occurs, and the horizontal stress is thereby raised by the wedge-like action of the portion of rock-mass descending in the manner known as normal faulting. The horizontal stress remains in this latter state only until solvent actions or possibly some other causes relieve it.

If the greatest horizontal stress is greater than the vertical stress or depth H it cannot be less than IH for any adjustment to occur. When slipping comes it tends to relieve the horizontal strain by the wedge-like motion of the two portions of the rock-mass approaching each other horizontally, and one of them being pushed up vertically in the manner known as reverse faulting.

Besides normal and reverse faults, we may have side-thrust faults, formed by the greatest and least principal stresses being both horizontal. The sliding is then horizontal and the fissure vertical. The San Francisco earthquake fissure-slip of 1906 was of this class, the greatest principal stress being about true north and south and the fissure running about N. 25° W.

We may also place in a separate class those faults in which none of the principal stresses are vertical.

We thus arrive at the following four classes of fissures:

Class.	Greatest Principal Stress.	Least Principal Stress.
Normal.	Vertical.	Horizontal.
Reverse.	Horizontal.	Vertical.
Side-thrust.	Horizontal.	Horizontal.
Skew.	Having no vertical axis of stress.	

Regions of Fissuring.

From the above classes of fissures we may divide the earth's crust into four regions, viz.:

A. The region of normal fissures.

B. The region of no fissures.

C. The region of side-thrust fissures.

D. The region of overthrust fissures.

Region *A* consists of rocks near the surface, where the solvent action of percolating waters or some more deep-seated disturbance makes normal fissures possible. This region is important as being the one we are most intimately associated with, and in which the richer concentrations of valuable mineral occur. In this region the least horizontal stress is not greater than $\frac{H}{I}$. If it is a little less, owing to there being no nearby fissure of proper angle, there may be a side-thrust fissure movement which really belongs to region *B*.

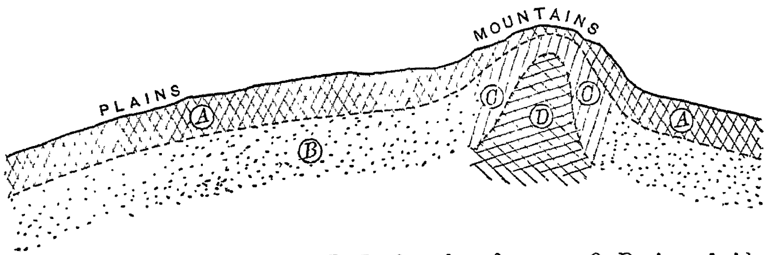
In regions *B* and *C* the least horizontal stress is more than $\frac{H}{I}$ and the greatest horizontal stress is less than IH. In region *B* the ratio of greatest to least horizontal stress is less than I, and in region *C* this ratio is greater than I. It is quite probable that there is no fissuring at all in regions which underlie the great plains to an enormous depth. Reasoning from this stand-point, it would seem that beneath the plains

region, A would correspond to a region of surface-water circulation. This deduction may throw some light on the much-disputed depth of action of surface-waters. Around the mountains we should expect some rocks to be subject to side-thrust movements, and the region B to give place to region C, which is of considerable importance in mining-districts.

In region D the greatest horizontal stress is not less than IH. This region becomes prominent near the earth's lines of weakness. The overthrust faults here force up large masses of rock into mountain ranges.

Skew fissures do not fall into any regional division because the requirement of a region is that the vertical stress shall be direct.

The diagram, Fig. 4, illustrates the probable distribution of the three regions and one extra sub-region in both plane and mountain country.



A. Region of normal fissures. *B*. Region of no fissures. *C*. Region of side-thrust fissures. *D*. Region of reverse fissures.

FIG. 4.—SHOWING DISTRIBUTION OF FISSURE-REGIONS.

Earthquakes.

Slipping is not a slow, creeping process, but a series of sudden jars. This is explained as follows:

During the process of slipping we have, in order to be exact, to consider the changes in the value of the coefficient of friction, α. Before motion actually starts we must use the statical coefficient, α_0, but during the process of slipping we must use the dynamical coefficient, α_1, which is less than α_0. The coefficient α_0 determines the value of the horizontal stress when motion starts, and α_1 determines its value when it stops, except that some allowance should be made for the extra stress which tends to arrest motion. The effect of the sudden slipping over a considerable distance is an earthquake, which is due solely to the difference in value of α_0 and α_1. In all

problems of fissure-equilibrium which we have considered by exact rules the statical coefficient of friction, α_0, is the only one we should take into consideration. The great horizontal stresses are no doubt formed by the arched condition of the crust and its relief from some of the support of the central core. This results from the diminution of the core by loss of heat.

Change of strain will be proportional to change of stress. Therefore, depth, superficial extent of fissure, elasticity of rock and other factors being equal, and α being 0.9, the violence of fissure-earthquakes in the region of reverse fissures will be 25 times as great as in the region of normal fissures. In the region of side-thrust fissuring, the violence will be from 5 to 25 times as great as in the region of normal fissures.

Influence of Third Axis.

When the third axis, y, with its corresponding principal stress, p_y, is taken into account we shall find that the maximum ratios of tangential to normal stress occur when $l = 0$ or $m = 0$ or $n = 0$, where l, m, n are the direction cosines of the normal to the plane under consideration. By comparison of these maximum values we find that if $p_x > p_y$ and $p_y > p_z$ the greatest maximums occur when $m = 0$. These are the cases we have examined in assuming the paper to be perpendicular to the y axis and, therefore, all the maximums we have so far considered are grand maximums; that is, they represent the most favorable conditions for jointing, fracturing, or slipping.

There are two special cases, however, which need our consideration, viz.:

$$\text{when } p_x > p_y = p_z \text{ and when } p_x = p_y > p_z$$

In either case the ellipsoid of stress becomes circular about what we may term the "odd" axis. The resultant fracture will make a definite angle with this odd axis, but there is no longer any reason why it should be definitely inclined to either of the other two axes, which we may call a "pair" of axes. This is the kind of fracture known as conchoidal. When there are small irregularities in the texture of the rock the plane of fissure will be somewhat modified, and when p_y is not exactly intermediate between p_x and p_z the fissure, although having the average strike and dip of a plane, will have conchoidal modifications locally displayed. The resultant fracture will make a more definite angle with the odd axis than with either of the other two, but the general average plane of the fissure will still include the intermediate or y axis.

Sub-Classes of Fissures.

We have previously considered only two principal axes of stress, viz., the axis of greatest stress and the axis of least stress. There is, however, of necessity, a third axis, which may have any stress value between the greatest and least stresses. The general strike and dip of a fracture are not affected by the

value of this intermediate principal stress, but when it is nearly equal to either of the other principal stresses, there becomes some question as to which of these two nearly equal stresses shall determine the direction, and the fracture seems to wabble to and from the mean course determined by the greatest and least stresses. If we call the "odd axis" the one which is least inclined to be equal in value to either of the others, which we may call a more or less perfect "pair," we may say that the resultant fracture will make a more definite angle with the odd axis than with either of the other two. This angle may, in fact, be more definite than though there were no axis appreciably odd. The surface of fracture which will be formed when a perfectly homogeneous medium is acted on by two exactly equal principal stresses and one odd one is that made up of elements of a cone on the odd axis. If, however, the intermediate axis is not exactly equal to one of the other axes, we may expect the fissure to tend towards a series of parallel corrugations whose length corresponds with the direction of slip. With a corrugated fissure it will not, in general, be possible to ascertain whether the greatest or the least axis of stress is the odd one; hence we cannot make as exact a classification as theory calls for. Unlike sinuous corrugations, conical surfaces of any considerable curvature can only form an extensive fissure by being linked together and forming a linked vein. With a linked vein there should, in general, be little difficulty in determining the odd axis.

The above cases may be classified as follows:

Plane fissures, . . .	no odd axis or pairing.
Corrugated fissures, }	{ imperfect pairing.
Linked fissures, . }	{ perfect pairing.

Secondary Displacement.

Secondary displacements may make an angle with the primary displacement and therefore with the corrugations, and so open up channels in which solutions can freely flow; even though the displacement is direct (that is, in the direction of corrugations formed at fracture), the wall-surfaces are more or less irregular along the corrugations, and channels are opened up.

Secondary-displacement "furrows" are often plowed up,

which gives us evidence of the direction of secondary displacements. This furrowing should not be mistaken for corrugation, which is usually formed on a larger scale than furrows. Fissures may thus be often divided, according to their secondary displacement, into:

Direct, . . . secondary displacement approximately along the corrugations.

Indirect, . . oblique secondary displacement at an angle with corrugations.

Square, . . . secondary displacement approximately at right angles to the corrugations.

Local Factors.

Displacement in an irregular fissure requires that the stress be confined to only a portion of the surface of the fissure, where it is correspondingly more intense than the mean stress; sub-jointing and sub-fissuring adjacent to the fissure may thus take place. At the same time, the grinding together of the walls takes off some of the irregularities, and supplies talc and other material to lubricate the movement of the fissure. All this gives rise to a more or less brecciated and pulverized filling-material between the two walls of the fissure, which, being now of conceivable breadth, we term a vein. The filling-material may be to a greater or less extent dissolved, replaced, and cemented together by mineral-bearing solutions, but this is a separate branch of the subject, the important mechanical consideration leading up to it being that a fissure arrived at the vein stage and filled with brecciated material offers the best conditions for the flow of solutions through deep-seated rocks. Under great stresses large open spaces cannot exist.

The pores of a close-grained, highly-compressed rock, such as is ordinarily met with in metal-mining regions, are very poor conveyors of mineral solutions, hence the interstitial spaces in brecciated material offer the only means by which such solutions may move about until conditions favorable to mineral deposition are reached.

In all cases, local weaknesses or irregularities are important factors in fixing the positions and extent of corrugations, furrows, and links.

Stratification or primary lamination not only makes the tan-

gential fracturing-stress different along different planes, but it also makes the elasticity of the rock different in different directions.

Sudden changes in the formation through which a vein is

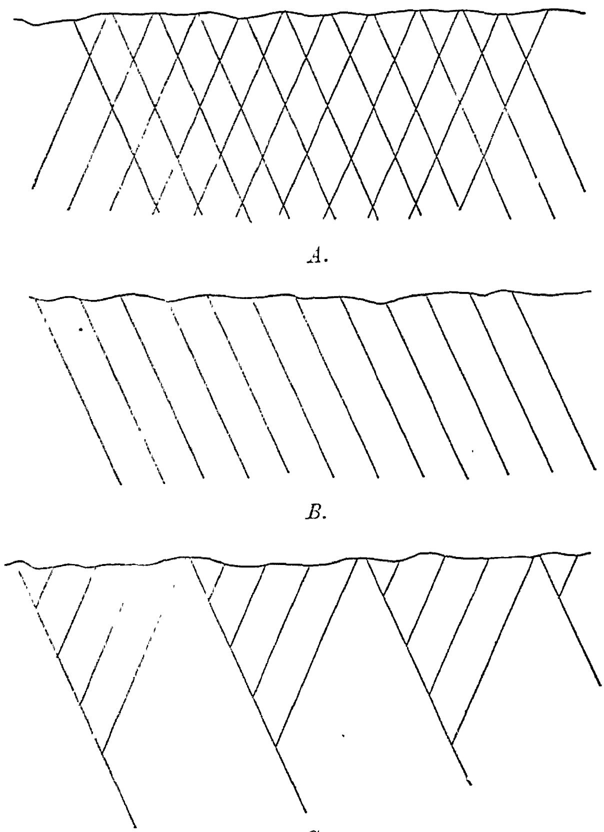


FIG. 5.—SYSTEMS OF FISSURES.

traveling are often noticed to be accompanied by equally sudden changes in the strike or dip of the vein.

Fissures in slates are sometimes a series of steps going first along and then across the formation.

Fissure-Systems.

The most complete system of fissures to be imagined as being formed by one condition of stress is a double system of parallel planes, inclined at $\cot^{-1} \alpha$ to one another, as shown in Fig. 5, *A*.

As, however, a displacement on any one of these planes would preclude subsequent displacements on the planes which intersect it, we are in general limited to a single set of parallel planes, as shown in Fig. 5, *B*, or such a system of meeting, but not intersecting, fissures as is shown in Fig. 5, *C*. Intersecting systems of fissures must be attributed to the existence of different conditions of stress at different times.

A classification of fissures is given in Table III. A vein of any class may fall into any sub-class, and similarly with secondary displacements and varieties, so that there are 108 separate divisions into which a fissure may be placed without specific measurements being made or cognizance taken of the filling-matter.

TABLE III.—*Classification of Fissures.*

Class.			Sub-Class.		Secondary Displacement.		Variety.	
Designation.	Greatest Principal Stress.	Least Principal Stress.	Designation.	Pairing of Axes.	Designation.	Angle with Primary Displacement.	Designation.	Depth of Formation.
Normal.	Vertical.	Horizontal.	Plane.	None.	Direct.	None or Small.	True.	Small.
Skew.	Inclined or horizontal.	Horizontal or inclined.	Corrugated.	Imperfect.	Oblique.	Some odd angle. The value of this angle may be designated in figures.	False-walled	Medium.
Over-thrust.	Horizontal.	Vertical.	Linked.	Perfect.				
Side-thrust.	Horizontal.	Horizontal.			Square.	Right angle or nearly so.	Shear-zone.	Great.

The further perusal of the "Laws of Fissures" may be advanced by taking the strains into consideration as well as the stresses. This may be the subject of another communication if there seems to be a demand for it.

Metal-Losses in Copper-Slags.

BY LEWIS T. WRIGHT, SAN FRANCISCO, CAL.

(New Haven Meeting, February, 1909.)

It is commonly believed by metallurgists that in copper-smelting, the copper in the slags, which is irreducible by continued "settling," is retained in the form of "prills" of matte. I have frequently held well-settled slag in a molten condition for a long period without being able thereby to reduce the copper-content. The slag acted as though it contained a minimum of dissolved or combined copper that could not be settled out by gravity. I have used reagents, but without satisfying myself in what form this copper existed. The same slags, by fine grinding and elutriation, could not be separated into portions containing more or less copper than the average content.

By treating copper-slags in an electric furnace a button of metallic iron and copper was produced, but the high temperature and the strong reducing-action of the furnace were influences that suggest an explanation of a result not obtainable with ordinary smelting-temperatures.

If all the copper in the slag were in the form of copper-matte, and not existing as a dissolved compound with some of the elements of the slag, then the other metals in the matte, such as gold and silver, should occur in the slag in the same ratio to the copper as to the copper in the accompanying matte.

This reasoning led me to study the ratio of metals in the products of smelting, and, apart from the issues discussed in this paper, the research has proved both interesting and illuminative.

On the assumption of a similar ratio of metals in matte and slag, if the percentage of copper in the slag accompanying a matte containing 50 oz. of silver and 1 oz. of gold per ton of copper was 0.3 per cent., or 96 oz. per ton of slag, there should be found 0.15 oz. of silver and 0.003 oz. of gold per ton of slag; but, in my experience, there is less silver and much

less gold in the slag than is required by the assumed law of similar metal-ratios.

In order to investigate this point, many accurate assays of matte and slag, covering long periods of time, were grouped into series so as to obtain an average effect of contemporaneousness of occurrence of both products, viz., matte and slag, and it was found that the greater the concentration of gold and silver in the copper-matte, the less, relatively, is the copper-gold-silver ratio in the slag.

Some of the results obtained, and which are now given in Table I., illustrate the general nature of the ratios thus discovered.

TABLE I.—*Gold- and Silver-Content of Copper-Matte and Accompanying Slag.*

Troy ounces of gold and silver per ton of copper.

Matte.		Slag.		Ratios.	
Gold.	Silver.	Gold.	Silver.	$\frac{\text{Gold in Slag.}}{\text{Gold in Matte.}}$	$\frac{\text{Silver in Slag.}}{\text{Silver in Matte.}}$
Ounces.	Ounces.	Ounces.	Ounces.		
27.9	62.8	8.65	48.9	0.31	0.78
3.5	33.2	2.3	30.9	0.66	0.93
2.5	166.4	1.62	127.8	0.65	0.76
3.04	152.0	2.10	116.0	0.69	0.76

It should be noted that in the above table the precious metals are not stated in the ordinary manner in ounces per ton of matte and slag respectively, but in ounces per ton of copper. Thus a 50-per cent. matte containing 13.95 oz. of gold per ton of matte is shown in the table as containing 27.9 oz. of gold. In the same manner, the slag stated in the table as containing 8.65 oz. of gold would, if it contained 0.3 per cent. of copper, in the ordinary manner be stated as containing 0.02595 oz. of gold per ton of slag.

If all the copper in the slag existed as matte entrained in the slag, the metal-ratio should be the same as in the matte; but this is not so in practice.

David Browne, of Sudbury, Ontario, informs me that the nickel-copper ratio of matte is not the same as that of the accompanying slag.

Although there is not enough data as to the form, or forms,

in which the irreducible copper exists in the slag to justify more than a suspension of judgment on this point, still the relations noted indicate that some portion of the metals is dissolved in the molten slag.

The reversible chemical reaction, $Cu_2S \leftrightarrows CuS + Cu$, explains the presence of free copper in solid copper-mattes, which is more marked in high-grade than in low-grade mattes. If free copper, also, exists in the molten matte and in increased quantity with the higher grade, the general observation that the copper carried away in well-settled slag increased with the

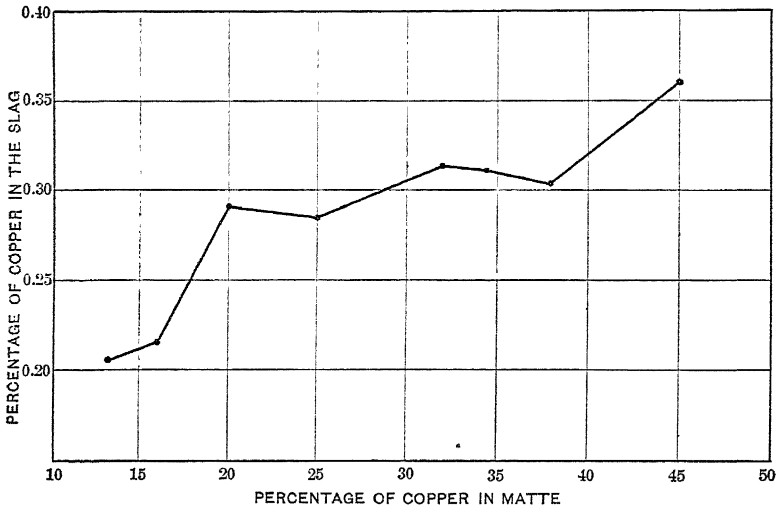


FIG. 1.—CURVE SHOWING VARIATION OF COPPER IN COPPER-MATTES AND ACCOMPANYING SLAGS.

grade of the accompanying matte is more suggestive of the influence of some physico-chemical property than of the fortuities of mechanical separation. Between the limits of 15- and 45-per cent. copper-matte the accompanying slags increase by an average of 0.005 per cent. of copper per 1 per cent. of increase of copper in the matte.

This increase, given in Fig. 1 as a curve, shows a well-marked flattening between 20- and 37-per cent. copper-matte, and depressions at 25- and 37-per cent. matte.

The distribution of a soluble body in two solvents is independent both of the relative amounts of the two solvents present and of the absolute concentration. The substance is distributed so that the ratio of its concentration in each solvent

is constant. If this so-called "law of distribution" were strictly applicable to copper-mattes and accompanying slags, and there were no interfering or modifying influences, then the ratios given in the last two columns of Table I. would be nearer to 1 than they are.

Other things being equal, a difference of 1 per cent. up of SiO_2 in the slag makes a regular difference of about 0.01 per cent. down in its copper-content. The regularity of this occurrence is more suggestive of solution than of differences due to more or less perfect settling.

If slag formed in making a matte having a certain metal-ratio be kept in molten contact with matte possessing a different metal-ratio, the slag will acquire a new metal-ratio due to that of the latter matte. The matte and slag appear to act as solvents and divide the metals accordingly.

If slag formed in making matte carrying a large quantity of precious metals be kept in molten contact with matte containing a small quantity of precious metals, the silver and gold will go out of the slag into the matte, although the percentage of copper in the slag, other things being equal, will remain constant. In this way the precious metals may be almost entirely removed from copper-slags.

I applied this knowledge of metal-ratios to great advantage in smelting copper-ores.

In a series of three reverberatory furnaces the middle one was built with a hearth lower than the others. All the slag from the outside furnaces is tapped through the middle furnace. The charge smelted in the outside furnaces yields a matte having high precious-metal values, while that in the middle and lower furnace yields a matte low in precious-metal values. In this way all the slag produced in either the outer or the inner furnaces can be discharged low in copper, gold, and silver.

The middle, or "cleaning," furnace smelted 6 per cent. more ore-charge than the two outer furnaces and consumed 17 per cent. less fuel, giving a total saving of 21.7 per cent. of fuel. The slag from the outer furnaces was not considered a part of the "ore-charge."

By using this method of smelting copper-ores the precious metals are recovered with an increased furnace-output and a large economy in fuel-consumption.

Hydraulic Dredging for Gold-Bearing Gravels.

BY HENRY G. GRANGER. CARTAGENA, COLOMBIA.

(New Haven Meeting, February, 1909.)

I. Introduction.

Repeated failures in attempts to work gold-bearing gravels by means of suction-dredges have created the impression that this method is impracticable. The suction-dredges have failed from three special causes: excessive wear and frequent breakage of pump-shell, runners and liners; inability to dredge compact gravel which would not readily move towards the intake; and, most important of all, closing of the suction-pipe by stones too large to enter it.

In my opinion, all three causes of trouble may be remedied: the first, by making stronger the parts most liable to wear and breakage; the second, by means for loosening the gravel at the intake; and the third, by sufficiently increasing the diameter of the suction-pipe. In suction gold-dredges, this diameter has not usually exceeded 10 or 12 in.; whereas there are few profitable gold-fields which can be considered dredgeable, in which stones more than 12 in. in diameter are not frequently encountered.

II. An Effective Suction-Dredge.

In harbor-operations, much larger stones are handled by suction-dredges. The large dredge of the Henry Steers Contracting Co. worked for nearly four years at League Island, Philadelphia, where it sucked up, passed through the pump, and forced through 3,700 ft. of discharge-pipe, gravel containing boulders up to 200 lb. in weight. This was done with no bad breaks and few shut-downs. The cutter and liners were the only parts liable to damage, and these were easily replaced with brief interruption of work. The chrome-steel liners are said to have lasted from six to nine months. I am informed that this dredge worked continously for nine

months at Harrison, N. J., on gravel, much of which carried stones as large as a man's head.

The essential features of this dredge are as follows:

The centrifugal pump, made by the Morris Machine Co., Baldwinsville, N. Y., was designed for a 22-in. suction, discharges at the bottom, and is driven at 225 rev. per min. Its shell is approximately 84 in. in diameter, with a runner about 60 in. in diameter. The runner-bearing has a water-sleeve to keep the sand out, supplied by a small force-pump worked at 60-lb. pressure. The pump is set 30 ft. from the bow. Power is supplied by a compound marine engine, with surface-condenser, 22 by 42 in. by 26-in. stroke, running 90 rev. per min. On the engine shaft is a 65-in. gear with wooden cogs, driving a 26-in. steel pinion. The wooden cogs are said to run about six months before renewal. On the pump-shaft keyed into the pinion is a slipping spring coupling. When a stone or log, too large to go through, gets into the shell, the runner is brought to a full stop without damage. This occasionally happens, and the obstacle may be removed by taking off the plate, with a loss of time averaging not more than 15 min. The pump uses from 450 to 500 h.p., which is considerably in excess of the economical requirement of a pump of this size, but with it the material may be forced through a pipe-line 4,000 ft. long. I am told that, with 1,000 ft. of discharge-pipe, 1,000 cu. yd. of gravel per hour may easily be handled. The total power used on the dredge is 800 horse-power.

The winch, with its various barrels for working-lines, suction- and cutter-hoisting lines and spud-hoisting lines, is a Lidgerwood machine, with two cylinders 14 by 18 in. The winch-engine also drives a chrome-steel gear, which through a shaft drives a 7-ft. chrome-steel cutter. In Fig. 1 a 6-ft. man is shown standing in the cutter. The cutter-shaft is connected with and driven by a universal joint. The wrought-iron suction-pipe, the mouth of which is just above the lower rim of the cutter, is connected by a flexible rubber joint to the cast-iron pipe which extends to the pump. The two timbers used as braces for the shaft and pipe are so hinged that their movement can only be vertical. The wrought-iron suction-pipe lasts for a year; the cast-iron pipe, including the elbow running from the rubber joint to the pump, lasts four years. The rubber elbow lasts from three months to a year.

This is said to be the largest suction-dredge in the world that has continously handled coarse gravel, and in capacity is the largest gravel-dredge of any class. The outboard suction is 54 ft. long, and the dredge can work to a depth of 30 ft. I am told that it has frequently sucked up from the bottom heavy spikes and tools which had been dropped into the water several feet in advance of the pipe.

The dredge is controlled by one man, standing behind a row of levers connected with the suction-hoist, various valves, etc. A vacuum-gauge shows the amount of material coming through the suction-pipe; a pressure-gauge shows whether the material is passing freely through the discharge-pipe; and a skilled

operator finds little difficulty in maintaining a fairly uniform movement, and as constant a ratio of gravel to water as can be secured with a bucket-dredge.

It seems safe to assume that, for the excavation and transportation of gold-bearing gravel, even containing large stones, this apparatus, provided with a cutter, to loosen such gravel as will not break up in the suction-pipe current, a centrifugal pump capable of producing a sufficient current, and a pipe large enough to pass all ordinary boulders, could be successfully employed.

In the original Bucyrus gold-dredge, the second lift was made by a centrifugal pump; and the same means, I believe, has been employed in dredging the phosphate-deposits of Florida.

III. Special Conditions of Gold-Dredging.

To adapt this design to gold-dredging, suitable means for the saving of gold must be provided. Where the metal is in heavy, shot-like particles, as at Bannock, Mont., its specific gravity will carry it quickly to the bottom of a sluice with deep water, carrying fairly coarse gravel; but where it is scaly and fine, like that found in the gravels of Oroville, Cal., in the Choco region of Colombia, and in most other gold-dredging fields, a current of water strong enough to move the gravel will carry with it practically all the gold. This was clearly shown at Oroville on the Continental dredge, where, as I am informed, until an elaborate system of undercurrents was introduced, not enough gold was saved in the sluices to pay for the trouble of cleaning up. In the Choco, operation of the dredges *Margaret* and *W. T. Curtis* proved that, with currents in the sluices strong enough to carry the gravel, virtually no gold was held in the riffles. The old hydraulic-mining sluices, in which grizzlies and undercurrents played such an important part, and the early pick-and-shovel telescope-boxes, used even now in some fields, show that, in order to save gold directly, the current must be below that required to move coarse gravel, since otherwise the gold will go with the gravel to the tail-race.

Indeed, the pay-streaks of gold-bearing placers and gulches, considered as natural sluices, tell the same story; for the gold which they contain has been transported by water, sometimes

to long distances from its source in the rocks, and deposited only when the current has been checked or slackened. On the other hand, when some such change causes deposition, that process (if not interrupted by extraordinary events like heavy floods) may go on for a long time at the spot thus designated by nature. In testing with the pan hundreds of pay-streaks, from 3 to 4 ft. or even more in thickness, carrying fine gold, I have found that, except for a few inches on top, they were generally of equal value to the bottom.

IV. History of Suction-Dredging in Gold-Bearing Gravel.

In this connection, it is unnecessary to repeat, or even to summarize, the data relative to the carrying-power of water, which are available in other forms. This phase of the subject has been covered in my paper on A Sea-Level Canal at Panama, presented at the present meeting of the Institute.[1] Much of the information therein presented has a direct bearing upon the problem of handling gold-bearing gravel by currents of high velocity. Theoretical conclusions on this point have been confirmed by experience also.

The following data, accompanied with references to the authorities cited, may serve to show the experience heretofore had with suction-dredges in gold-bearing gravels, and to guide the reader in more detailed inquiry:

1. At Alexandra, New Zealand, in 1887, a Welman suction-dredge was put in operation, and others were built later along the West Coast ocean beaches. They were found capable of dealing with fine gravel, sand, and shingle, but were not adapted to coarse gravel and large stones. For this reason they did not come into permanent use.[2]

2. In the neighborhood of Waipapa, Otago, New Zealand, suction-dredges have been found suitable for handling sand and small stones.[3] Concerning this type, T. A. Rickard wrote in an able paper on Alluvial Mining in Otago:[4]

"For irregular bottoms, the suction-pump dredge, of which the Welman is a good example, will be found best adapted. In this case, a powerful centrifugal

---

[1] *Bulletin* No. 25, January, 1909, pp. 1 to 37.

[2] *New Zealand Mining Handbook*, p. 245 (1906).

[3] *Report on the Mining Industries of New Zealand*, pp. 75 to 76 (1891).

[4] *Trans.*, xxi., 472 (1892–3).

pump draws up the water, gravel and gold, delivering them to the level of the tables. At Waipapa, stones 35 to 40 pounds in weight have been sent up by the pump; and it only requires an improvement in construction, giving durability and strength, to render it a most effective machine for this class of work."

3. At Six-mile Beach, on the ocean beach, about 18 miles from Fortrose, towards Waikawa, New Zealand, a dredging-plant was in operation (1890), consisting of a boat carrying a Welman dredge, and a centrifugal suction-pump, 3 ft. 6 in. in diameter, with a 13-in. delivery-pipe, and a velocity of 260 rev. per min. The swinging suction-pipe had a radius of 40 ft. and was fitted with a sleeve-nozzle and revolving cutter. The gold-saving tables were 64 ft. (?) wide, and had a fall of 1.5 in. per ft. Side-boxes collected the tailings and carried them to the stern of the dredge, where they were discharged through a 15-in. pipe, with a universal-socket joint, suspended from a post-crane. The material handled was mostly sand. Stones were caught on a hopper-plate and thrown, on either side, upon ground already dredged. The sand passed through the perforated plate into the hopper, and was carried by the water to the tables, upon which it was equably distributed. The gold, which is extremely fine, was caught in plush mats. The results were reported to be satisfactory.[5]

4. At Lake Brunton, Otago, there was a dredging-plant (1890) generally similar to the one at Six-mile Beach last described, but with a pump capable of maintaining a current which will easily carry and pass stones up to 56 lb. in weight. Large stones caught by the arms of the runner were broken up without injury to the centrifugal pump. Still another dredging-plant of this type, on Waipori Flat, proved successful in wet ground with no fall for draining or hydraulic sluicing.[6]

5. According to Thomas Egleston, in his paper entitled The Treatment of Fine Gold in the Sands of Snake River, Idaho, much of the gold in the gravel of Snake river had for years been recovered by the simple method of gravity-concentration on burlap tables after the fine material had been separated from the coarse gravel by passing through a screen-floored sluice-box.[7] In 1904, Robert H. Bell, State Inspector of Mines,

---

[5] *Otago Witness* (April 23, 1890), in which additional details are given.

[6] *Otago Witness* (April 23, 1890).

[7] *Trans.*, xviii., 597 (1889–90).

expressed the conclusion that the suction-dredge would solve the problem of profitably working on a large scale the broad, flat bars of Snake river. Mr. Bell says[8] the saving of the gold presents no further difficulty, but what is needed is a method of handling large quantities of the low-grade material. His opinion is based in part upon the experience of the Sweetser-Burroughs Mining Co., which, in 1894, introduced a floating suction-dredge on Snake river. It was started with a 6-in. intake-nozzle, afterwards increased to 10 in. The pump was designed to pass any stone which might reach it through the suction-pipe: and the plant had been in successful operation, with successive improvements in strength and stability, ever since its first installation, up to the date of Mr. Bell's interesting and detailed account of it, in 1904. In that report he said:

"The cost of handling gravel at this plant, including all charges, is 4 1-2 cents per cubic yard. Working in the river bed, most of the gravel being raised from below the water surface, a good deal of the material handled runs from 10 to 20 cents per cubic yard, and affords a handsome margin of profit."

Unfortunately, Mr. Bell was obliged to report in 1906[9] that this company, having exhausted the most favorable ground available for its operations, skimming the best layers of gravel to the depth of about 6 ft., with an average yield of something less than 10 cents per cu. yd., and at a considerable profit, had suspended operations.

Mr. Bell's description of the Snake river gold is worthy of special notice. He says, in his report of 1906, already cited:

"The Snake river's fine gold is finer than any natural placer gold I know of. It is high grade and worth $19.50 per ounce, but requires fully 1,500 colors to weigh one cent in value; yet under a powerful microscope, each color is an individual nugget showing abrasion marks. These particles are often coated, touched or spotted with a crystalline white film of some foreign substance that looks like silica under the glass, and this is what makes it necessary to polish it in a grinding pan before it will amalgamate freely."

If such gold as that can be recovered to the extent of 95 per cent., as given by Mr. Bell, from the material delivered, we may well accept his conclusion, that the real remaining economical problem is the cheap handling of the material.

---

[8] *Report of the State Inspector of Mines of Idaho*, p. 48 *et seq.* (1904).

[9] *Eighth Annual Report of the State Inspector of Mines of Idaho*, p. 114 (1906).

6. On the Sacramento river, California, the operations of the Huron Submarine Mining & Construction Co. with a suction-dredge are peculiarly interesting. They are carried on in "blue ground" from 8 to 25 ft. thick, and containing many boulders. The gold is coarse and well worn; the bed-rock is igneous and very rough. The dredge is unlike any other ever built. The boat is 65 ft. long, 24 ft. wide, and draws 2 ft. A 75-h.p. engine operates a rock-pump, an air-compressor, and auxiliary machinery. In the middle of the boat is a steel shaft made in sections for extension to any required depth. These sections are boat-shaped, measuring in horizontal cross-section 11.5 by 8 ft. The vertical length of each section is 6 ft. The sections of the shaft are set with the sharp edge up stream, to diminish the resistance offered to the current. The lower sections are cylindrical, and the lowest is provided with a water-tight door for ingress and egress of divers. This shaft is sunk through water and gravel to bed-rock. Down through it extends the 10-in. column of the rock-pump, and a 2-in. hose to carry water at 100-lb. pressure to the mouth of the pump-column. A diver in the shaft, hose in hand, works with freedom, directing gravel to the mouth of the suction-line, and cleaning the gold from the bed-rock crevices. The capacity of the pump is 1,500 cu. yd. of gravel per day, with water enough to wash it in the sluices. A diver works continuously under water 5 hr. a day. The estimated cost of dredging is 3 cents per cubic yard.[10]

7. Earlier records of California experience are given in the special volume[11] issued in 1899 by the California Miners' Association. R. H. Postlethwaite, in an article on Dredging for Gold, sums up as follows (p. 93) the situation up to that date concerning the suction-dredge.

> "The hydraulic-suction dredge has, however, in times past, had its supporters, and immense sums have been expended on it generally with very unsatisfactory results, although there are a few cases where it can be made to work with advantage. For example, in digging sand and conveying it long distances it probably has no equal; but for lifting heavy gravel and boulders, and for the picking up of gold, it has of necessity generally proved a failure, except in very rich ground."

As to the picking-up of gold, mentioned by Mr. Postlethwaite,

[10] *Bulletin* No. 36, *California State Mining Bureau* (1905).

[11] *California Mines and Minerals*, published by the California Miners' Association, under the direction of Edward H. Benjamin, Secretary for the California Meeting of the American Institute of Mining Engineers (San Francisco, 1899).

I think that the failure of a suction-dredge to do this must be due to bad construction, or bad management, or both. Practical experiment, confirming all standard formulas, will show that a suction-pipe, especially if provided with a hood, and drawing its water from the bottom, can clean the bed-rock of all loose gold, including ordinary nuggets not actually wedged in very hard and solid material. B. K. Morse, a member of the Institute, informs me that, in 1890, a dredge with 20-in. suction, used by the N. Y. Shipbuilding Co. in operations for reclaiming land at Camden, N. J., frequently pumped up coins (sometimes of gold), and occasionally old Revolutionary bullets and cannon-balls, as well as innumerable pieces of iron of various forms. and once an entire musket, so that the workmen were always on the look-out for "curios."

An article in the same volume (p. 434) by Thomas J. Barbour, on The Evans Hydraulic Gravel Elevator, describes an apparatus developed in New Zealand, and used in hydraulic sluicing, as described below. This article contains valuable data of experience as to the moving of material by water through pipes.

§. In Australia, hydraulic sluicing with centrifugal pumps, which should not be confounded with suction-dredging of primary material, is much used in reworking the old alluvial diggings, and has been found profitable. This method has been described under the title, Hydraulic Dredging in Australia, by F. Danvers Power.[12] While it is not directly a case of hydraulic dredging proper, it furnishes additional confirmation as to the possibility of transporting material by means of water-currents such as the centrifugal pump can produce. In the Beechworth and Castlemaine districts (especially the latter), many plants are reported as using this method of hydraulic sluicing. Work is begun by excavating and filling with water a hole 60 ft. square and 6 ft. deep, in which the barge or scow is floated. The required head of 60 to 70 lb. water-pressure is obtained from natural sources or supplied by a 12-in. centrifugal pump. The average yield of the old placer-ground (about 14 ft. deep) in the Castlemaine district has been nearly 230 oz. gold per acre. It is estimated that a yield of 125 oz. per acre would yield a profit; and it is reported that

---

[12] *Engineering and Mining Journal*, vol. lxxxi., No. 16, pp. 759 to 761 (Apr. 21, 1906).

the Castlemaine Junction Sluicing Co., after 15 months' work, returned to its stockholders the whole of the capital originally invested, with a substantial extra dividend besides. Further details of this interesting method are given by Mr. Power in the article cited above. I will note here, as pertinent to my subject, only his report that the gravel-pumps used raise and pass boulders up to about 60 lb. in weight.

V. Conclusions as to Suction-Dredging.

The facts above summarized and other information readily accessible to the student of dredging-problems warrant the following conclusions:

1. The problem of gold-extraction may be, and should be, separated from that of the cheap excavation and handling of the gold-bearing material. Each can be best solved by itself, after the elimination of all the conditions which properly belong only to the other. Dredging must not be made slower or dearer for fear of a loss of gold in subsequent operations; and failure of technical efficiency in gold-extraction should not be chargeable to the method of original excavation.

2. The separation having been made, cheap and rapid dredging on a large scale is evidently a prime requisite for the exploitation of gold-bearing gravels.

3. Where the quantity of auriferous gravel controlled is sufficient to warrant a large investment, the hydraulic suction-dredge will give rapid and economical results. Such conditions are found in the newer fields of California, along the rivers of Brazil, in some of the rivers and flats of Dutch Guiana, possibly in parts of Siberia, and certainly in the large streams of Colombia, especially in the Choco district.

The greatest expense of gold-dredging, as at present practiced, is the repair of the bucket-line, including the tumblers. *Bulletin No.* 36 (1905) *of the California State Mining Bureau* (Table 5), shows these repairs amounted to twice as much as all others put together, and more than one-third of the entire operating-expense. In another place, 39.3 per cent. of the time lost in stoppages by a given machine is reported to have been due to the same cause. Judging by the record of the Steers dredge, we may assert that a suction-dredge would offer in this particular a large saving of both time and money,

to say nothing of first cost, which is, for such a dredge, by no means proportioned to its capacity.

As to running-expenses, I am informed that the daily cost of operating a Steers dredge, such as has been described above, will not average over $250 per day, including interest and sinking-fund, and that its capacity of 1,000 cu. yd. of gravel per hr. could safely be maintained for 15 hr. in the 24, if the gravel did not have to be spread from a pipe. In my judgment, the allowance of 9 hr. in the 24 would cover all necessary stoppages for removing snags and large stones, and for other reasons common to all dredging-operations. Assuming the gold-saving and stacking features to cost the same as in the method of piping astern and spreading, this would give a total cost of about 1.67 cents per cu. yd., with the important additional advantage of an unprecedented daily yield.

This paper proposes, not a change of practice in the sense of the adoption of a new principle of dredging gold-bearing gravel, but a change of practice, in so far as it involves the re-adoption of a well-known principle, with such improvements in apparatus as will give it fair play. Suction-dredges have made already creditable records here and there; but they have often failed to give satisfaction or to pay dividends, by reason of their imperfect construction or inadequate dimensions, or of local conditions for which they could not fairly be held responsible. In short, the suction-dredge has not had, in this field, an opportunity to show what it could do at its best; and from my own experience I think the general tendency is to persevere in experiments with various forms of bucket-dredges, which (under certain adverse conditions only too likely to occur) are bound to fail.

What I propose, with the Steers dredge as a basis, is a suction-dredge of maximum durability and efficiency. In comparison with other mining-machinery, the gold-dredge has hardly maintained its due proportion of progress. Within the memory of many still living, the stamp-mill has been developed from the primitive Cornish battery, with a daily capacity per stamp of 500 to 600 lb. of ore crushed through an 11-mesh screen (or its German prototype and companion), to the modern plant with a daily capacity per head of 4 tons crushed through 40-mesh—an increase of 36 fold in efficiency. Within the

period of a generation, the capacity of river- and harbor-dredges has increased from 800 cu. yd. per day to 6,000 cu. yd. per hour—a 75-fold increase. In gold-dredging, on the other hand, the most recent 13-ft. Bucyrus apparatus has shown a capacity not more than five times that of the original New Zealand dredges of 20 years ago.

The question of profitable gold-dredging depends on the first cost, capacity and operating-expense of the plant, as against the quantity and value of the gravel to be moved.

At the present time, I would regard a dredge having a capacity of 1,000 cu. yd. per hr. as a sufficient step in advance; and I offer the specifications for such a dredge. If for use in California, where commercial electricity is available, or where water under natural head can be employed, as in the latest New Zealand practice, my specifications would require modification in several items. Being under the necessity of assuming the conditions to be met, I have taken those which permit the boat to be built on tide-water at or near New York, towed to her field of work, and there operated with wood-gas.

VI. Specifications for a Hydraulic Gold-Dredge.

Power.

The maximum power-requirements of a gold-dredge are estimated from the Steers dredge in operation, and proportioned according to the Oroville practice with dredges having a nominal capacity of 100 cu. yd. per hr. On this basis, the power-requirement would be:

Power Required for Dredge Moving 1,000 *Cu. Yd. of Gravel Per Hour.*

	H.P.
Pump,	800
Cutter,	60
Winch,	80
Screen,	100
Stacker,	90
Bank-pump, primer, and clean-up,	50
Shop-drive,	10
Primer-pump,	10
Force-pump for runner-bearing,	10
Electric light,	15
Total,	1,225

Pump.

In the opinion of Carl Lager, Superintendent of the Morris Machine Works, Baldwinsville, N. Y., which I adopt, with appreciation of his competent advice, a dredge intended to handle continuously gravel that would enter a 24-in. pipe would require a pump of cast steel, heavily reinforced and provided with manganese- or chrome-steel liners. The shell would need to be about 12 ft. in diameter by 4 ft. in width. The runner would be 7 ft. in diameter and make 170 rev. per min. The runner-shaft would have a slip-coupling, so that the runner might be stopped short by an obstruction without anything breaking. For dredging to a depth of 60 ft. the economical velocity of flow in a 24-in. pipe would be 10 ft. per sec., giving a flow of 14,200 gal. per min. If 23 per cent. of this flow was gravel, it would give the required 1,000 cu. yd. per hour. In practice, a skillful lever-man, working in free gravel, should raise 25 per cent. continuously. The outboard suction-pipe would be about 90 ft. long. For the less than 200 ft. of pipe required, the friction-head, at the estimated velocity of 10 ft. per sec., would, apparently, be less than 10 ft.; but, for safety, the total friction-head was assumed at 60 ft. and the pump-efficiency at 40 per cent., giving an estimated power-requirement of 600 h.p. delivered on the runner-shaft. The pump would discharge at one side of the bottom, and the discharge-pipe, before curving to the sluices, would have a removable section for taking out gravel in the event of a shut-down under load. The pump would weigh about 65,000 lb., and cost not more than $10,000. For a reserve set of liners, $2,000 more may be allowed; and for an extra runner, $1,500.

Cutter.

The cutter, to provide against breaking when stalled, should be driven by a 9-in. chrome-steel shaft, which should be supported by the trussed suction-pipe. Cast-steel pipe would probably be best for continuous gravel-suction. The suction-pipe and cutter, with drive and gears, would weigh about 70,000 lb., and cost about $9,000. The curve of the cutter-blades should be such as to leave the mouth of the pipe but a few inches above bed-rock when dredging at the full depth of 60 ft. The diameter of the cutter should be, say, 7 feet.

The Steers dredge has two large timbers as stiffeners to the suction-pipe and cutter-shaft, as shown in Fig. 1. It would be well to use, instead of these, two riveted hydraulic pipes, 24 in. in diameter, which would give a net buoyant effect of about

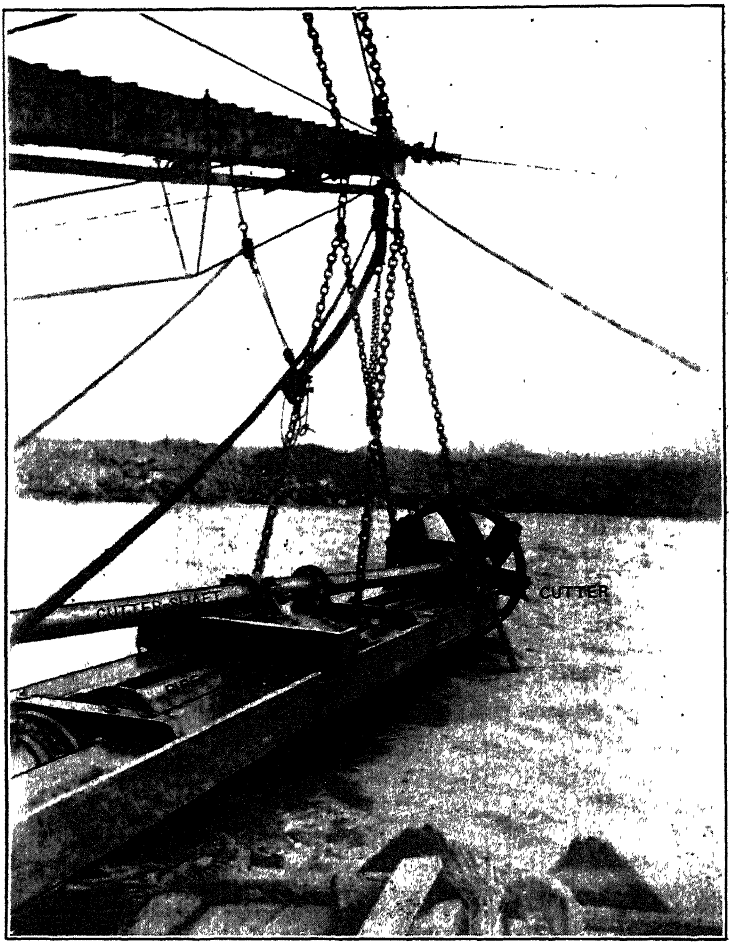


Fig. 1.—Cutter and Suction-Pipe of the Steers Dredge.

326 lb. per running foot, with the further advantage that the lengths of supporting pipe could be made exactly equal to the lengths of the cutter-shaft and suction-pipe, and, when desired, the whole ladder could be materially shortened or lengthened. These pipes and the cutter-shaft could be carried with advan-

tage in a lattice truss, with a removable section, to permit permanent use at shallow depths.

Winch.

Owing to the difficulty on the Steers dredge of throwing the cutter in and out of gear, it seems advisable to have the winches entirely separate. They should be very powerful machines, and, for gold-dredging in flowing rivers, would be found more satisfactory if divided into two units, one on each side of the dredge, each with a spool projecting, for warping, handling lines, etc. There should be 12 winch-barrels, of sizes proportioned to their respective services: two for head-lines (one in reserve), which would be built to carry each 2,000 ft. of heavy steel cables; one for ladder-hoist; three for operating a snag-and-rock grab; two for swinging-lines; two for stern-lines; and two for the spuds.

The winches should have gearing for two speeds: a very slow one for difficult situations, and a relatively fast one for ordinary circumstances. Extra lines would be an important part of this equipment. The total for this winch may be estimated at about $50,000, and a weight of 100,000 lb. should be allowed.

Bank-Pump and Clean-Up.

It will frequently be found desirable to cave the bank ahead of the dredge. The nozzle could be attached to the snag-boom, and handled close to the bank. The same pump would also answer as a primer, and to supply the amalgam-barrels and clean-up tanks; 50 h.p. would fill this requirement. A weight of 10,000 lb. and a cost of about $1,500 should be allowed for this item.

Force-Pump.

The force-pump to supply water under pressure to the main runner-bearing in the dredging-pump, would have to be running both before and after the dredging-pump was started, as well as during dredging; 10 h.p. would cover this item, with about 700 lb. weight and a cost of $600.

Electric Light.

The dredge should be lighted throughout by electricity, with incandescent lamps handy to the machinery, two arcs at

the bow, two arcs over tables, and one in the engine-room. Flaming-arc lamps should be used. A search-light should also be placed over the pilot-house, to be switched in when desired; 15 h.p. should cover this item, with a cost of $1,000 and a weight of 3,000 pounds.

Shop.

In connection with a large forge and blacksmith equipment, the dredge should have a lathe, shears, radial drill, and milling-machine; also a traveler for handling the plates that would have to be constantly ready for the screen. It would be advisable to do this work on board, at least until the screen-requirements of a given district were standardized; 10 h.p. and a cost of $3,000, with a weight of 10,000 lb., should cover this item.

Stacker.

The capacity of the stacker should be such as to be able to handle a maximum of 10 cu. yd. per min.; length, say, 89 ft. A belt, running at not over 300 ft. per min., would seem to be the best to handle the gravel, without bringing too heavy a weight on the stern. A main belt, 60 in. wide, with the sides gently curved by idlers for 18 in. from the edges, and a superimposed flat belt to take the wear, should answer the purpose; 90 h.p., a weight of 40,000 lb., and a cost of $10,000, should cover this feature.

Screen.

In view of the screen-areas of present dredges, and the record of the Sweetser-Burroughs dredge, quoted above, a screen with a surface of 2,000 sq. ft. would apparently be able to handle 1,000 cu. yd. per hr. The pipe would discharge, at 26 to 30 ft. above the water-level, into a stationary sluice, 25 by 20 ft. in size, with a bottom of 0.25-in. plate-steel, set at a grade of 0.25 in. to the foot and perforated with $\frac{1}{8}$-in. holes, countersunk on the under side. This stationary screen would discharge, in turn, on to a shaking-screen of the same width extending over the rest of the length of the tables. The holes would increase in diameter to 0.5 in. at the lower end of the screen, and should be close enough together to make sure that all the water will drain through them before the gravel reaches the lower end. The shaking-screen would be set at a

grade of 1.5 in. per ft., with an adjustable hanger for reducing the grade as desired, as shown in Fig. 2.

Gold-Saving Areas.

Dredges now in use have a saving-area of from 750 to 1,000 sq. ft. for a capacity of 100 cu. yd. of gravel per hour. This ratio should be at least maintained, and increased as much as practicable. It seems quite feasible to get a satisfactory area. The old *Plutus* and *California*, the first dredges in Oroville to use the shaking-screen (both steam-shovel dredges and long since abandoned), used a high drop of 2 ft. or more on to the tables, with the idea that the gold that fell with this impact would stay and amalgamate. But Vail, a veteran gold-saver, argued that the gold which would stay by reason of its weight would stay anyhow, and that by utilizing this space for a riffle-board, having shallow auger-holes, he could provide a greater area to amalgamate the fine gold. His boards, set zigzag from the upper to the lower level, were successful; and, somewhat modified, they are used in some of the latest dredges. The following is the suggested gold-saving area for the dredge here proposed:

	Sq. Ft.
4 sets of boards, each 2,000 sq. ft., ...	8,000
Table-surface, 40 by 100 ft., ...	4,000
Rose-box at table-discharge, 100, at 10 sq. ft. each, . .	1,000
Sluices, at least 500 sq. ft. each, ...	1,000
Total, ...	14,000

Hull.

For a field where the dredge would never have to move out of the reach of commercial electricity, and would not be compelled to carry long lines to work or warp against strong currents, the pump could be set in the center of the dredge, a much shorter hull could be used, and greater table-area secured by increased width. But for use in foreign river-bottoms, a hull 175 by 60 by 8 ft. would be better. The bow should be tapered and have a gentle rise for, say, 20 ft., and the stern should have an equal curve for the "get-away." The importance of the stern curve is sometimes not fully understood. Church has clearly shown that a blunt stern on a scow is a greater strain on a tow-line than a blunt bow, on account of

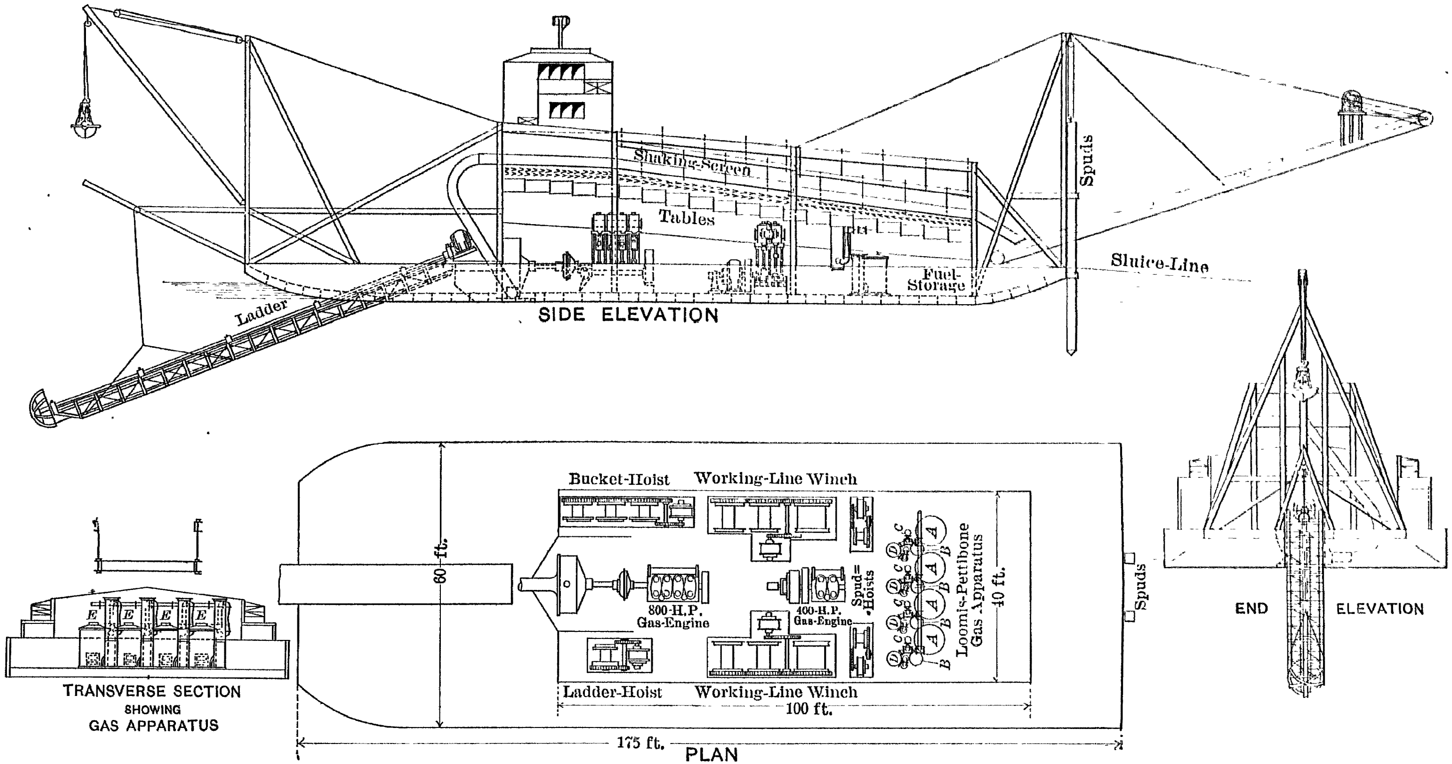


Fig. 2.—Plan, Elevations, and Section of the Proposed Granger Dredge.

the great mass of water, the momentum of which it has to overcome.

Properly constructed of steel, with suitable wood-lined quarters for manager, captain, and crew, and adequate strength in every part for a sea-voyage, as well as for the strain of regular operation, the hull ought not to cost more than $45,000.

Fuel.

In New Zealand and elsewhere, dredges are operated with steam, using coal as fuel. But gold-dredges in South America would have to depend, at least for some time to come, on wood as fuel. Those who have been compelled to make steam with comparatively fresh-cut wood, wet by rain or by transportation in canoes, will probably agree with me that it is unsafe to estimate on maintaining 120 h.p. on less than $\frac{1}{3}$ cord per hr., or 8 cords per 24 hr. At this rate, 1,200 h.p. for the proposed dredge would require 80 cords per day, which, under ordinary circumstances, would be a prohibitory condition.

The paper of Mr. Langton on The Power Plant of the Moctezuma Copper Co. at Nacozari, Sonora, Mexico,[13] raises the interesting question of the use of gas-engines driven with gas from wood.

I am inclined to think that, under suitable circumstances, this means of generating power might be found practicable and profitable in some regions such as I have mentioned. But the source of power is a question of locality. In many places it would be best to utilize natural water-power through the electric current. For the proposed dredge, when electricity is not available, I suggest a marine producer-gas plant, as shown in Fig. 2, which, according to the Loomis-Pettibone Engineering Co., will deliver the required power with a wood-consumption of only 17 cords per day. The cost of such a plant would be about $12,000.

Engines.

Three gas-engines would be required: one for the dredging-pump, with four 20- by 25-in. cylinders, guaranteed to develop 700 h.p. at 170 rev. per min., and able to stand up under 1,000 h.p., though with somewhat less economy. This engine, includ-

---

[13] *Trans.*, xxxiv., 748 to 776 (1904).

ing all auxiliaries, air-compressor, circulating-pump, etc., would weigh 40,000 lb., and cost $30,800.

The dynamo to distribute power to the various motors would be run by an engine with two cylinders (of the same dimensions, so that one repair set only need be kept), weighing 20,000 lb., and costing $15,400. This cost is stated to cover an extra cylinder set, extra one-half crank-shaft of total weight of 6,500 lb., and a 14 h.p. donkey-engine of two cylinders, with dynamo for lighting and pump, as well as a compressor for filling the air-tank for starting the large engines.

Crew and Expense.

For continuous operation and the occasional handling of heavy lines, the following crew should fill the requirements at the stated daily wage, plus food:

Per Day.	
1 general manager (who should be an experienced gold-saver),	$10.00
1 captain, with experience in handling dredges in strong currents,	5.00
1 chief engineer, gas and electric,	5.00
1 machinist and blacksmith,	5.00
2 cooks, at $1.00,	2.00
6	$27.00
Per Shift.	
2 levermen (experienced in gravel-pumping), at $4.00,	$8.00
1 engineer, gas and electric,	4.00
1 engineer assistant,	3.00
2 oilers, at $2.00,	4.00
1 boss fireman,	2.00
2 assistant firemen, at $1.00,	2.00
4 deckhands, at $1.00,	4.00
13	$27.00
Or for three shifts,	81.00
Total wages,	$108.00
Food for 45, at $1.00,	45.00
Repairs and supplies,	50.00
Wood, maximum 17 cords, at not over $3.00, delivered,	51.00
Total daily cost,	$254.00

The allowance for food and wood is liberal, and a smaller crew would perhaps suffice; but these figures should not be cut for preliminary estimates.

Capacity and Cost Per Yard.

With ordinary care in painting and keeping up, a dredge built in the manner suggested should last many years. With all the machinery of the best character, a full supply of duplicates for all possible contingencies, a good crew and good management, the shut-downs should be but slight, outside of encountering snags and sunken logs, to handle which this hydraulic dredge would be as well equipped as any other. Very few stones would be found in the average dredging-field that would not go through a 24-in. ring, so there should be little time lost on this score. It seems entirely reasonable to expect to dredge 20 hr. per day. This, at 1,000 yd. per hr., would give a daily capacity of 20,000 cu. yd. of gravel.

Notwithstanding the smaller sum of the component items of cost given above, it would not be prudent to reckon that the cost of such a dredge, by the time it has been set up in the port of its construction, given a trial run, then housed and towed to its destination, and put at work on the ground, would be less than from $200,000 to $250,000. It should be remembered, however, that a dredge of this capacity and of the table-area to be used, and to employ commercial electricity as power, would cost less than half this sum, or no more than a good 7-ft. chain-bucket dredge.

A sinking fund, to retire $200,000 in 10 years, would require $83.29 per $1,000 per year, or per day (at 300 days per year), $55.53. This, added to the daily running-expense of $254, with a further allowance of $25 for general expense, gives $335 as the daily grand total; and this sum, divided by 20,000 cu. yd. of gravel handled daily, gives $0.01675 as the cost per cubic yard.

Even if the ground should be unusually full of logs, it is evident that considerable time could be lost without making the cost per yard excessive. In fact, the power of the dredge is so calculated that more than 1,250 cu. yd. of free gravel could be readily dredged and screened per hour, so that the assumed average of 1,000 cu. yd. per hr. can easily be maintained.

VII. Conclusion.

The design and specifications submitted in this paper are not offered as satisfying all conditions of gold-dredging. Many

rich stream-beds would not give "elbow-room" for such a dredge: many otherwise favorable gold-bearing areas are too small to warrant the cost of its installation; and, in some localities, as I have already remarked, the supply of adequate power might be, at the present time, a matter of economically insuperable difficulty. But I believe that, under suitable conditions, the use of hydraulic suction-dredges of a minimum capacity of 1,000 cu. yd. per hr. would be exceptionally profitable; and I hope that my suggestions will elicit the discussion by engineers in this department of a proposition which, to my mind, promises so great an advance in this relatively new form of mining industry.

Several manufacturing concerns have been mentioned, as a matter of record, in the foregoing pages; but I need hardly say that these references are not intended either to advertise particular parties, or to convey the impression that these parties only would be able to furnish the machinery contemplated. On the contrary, there is nothing in the design here proposed which could not be constructed by any establishment experienced in such work.

The Treatment of Slime on Vanners.

BY RUDOLF GAHL, PH.D., MORENCI, ARIZ.

(New Haven Meeting, February, 1909.)

SOME time ago the Detroit Copper Mining Co. had to decide the question whether it would pay to re-treat slime-tailings, and several machines were tested in order to ascertain the type of construction which would give the greatest saving. In previous tests on the ore of this company, several tables proved far inferior to the Frue vanner, and for this reason they did not receive any serious consideration. In the late tests, however, one table was found to surpass the vanner in the percentage of saving made.

The Frue vanner used in these tests had been adjusted approximately to the slope which the slime-vanners in the concentrator of the Detroit Copper Co. had at that time, about 2 in. between the inside of the wooden posts, or 0.272 in. per ft. It had not been determined, however, if this slope was the most economical one for treating the slime-feed in the mill, or for re-treating the tailings from the slime-vanners, and I was instructed to ascertain by test-runs, under different conditions, the best adjustments for the work to be done.

In order to determine if a change in the adjustments of the machines would improve the results, it seemed sufficient to continue running the tests between the vanner and the table before mentioned, keeping the feed and all other conditions the same, but varying the adjustments of the vanner from run to run.

I employed a vanner-man of long experience to operate these machines, who expressed the opinion that a slope between 2⅜ and 2.5 in. between the posts (0.323 and 0.340 in. per ft.) would give the best results.

Some runs made this way gave a considerable improvement. While for a slope of 2 in. (0.272 in. per ft.) the saving had been only 40.5 per cent. of that of the table, it was 49.2 per cent. for 2.5 in. (0.340 in. per ft.) slope. A decrease of

the slope to 1.5 in. (0.204 in. per ft.) decreased the saving to 32.7 per cent. of that of the table. So far, these tests had been made with a smooth belt and with 224 (1-in.) side-strokes per minute. A reduction in the number of side-strokes from 224 to 188 increased the saving several per cent., and an increase of the slope to 3 in. (0.408 in. per ft.) added 8 per cent. to the saving.

An old corrugated belt was then tried, starting with 2.5. in. slope (0.340 in. per ft.). The saving was 78 per cent. of the saving of the table; for 3.5 in. (0.476 in. per ft.) it was 97 per cent., and for 4.5 in. (0.612 in. per ft.) the vanner made a better saving than the table. A decrease in the number of side-strokes slightly increased the saving.

These results could be interpreted in two ways. It was possible that the pulp, which had once gone over a vanner with a somewhat flat smooth belt (all vanners in the concentrator had smooth belts at that time), was not fit for re-treatment on a machine with practically the same adjustments. To take out mineral that could not be saved on the first vanner, either the smooth belt might have to be set steep, or a corrugated belt might have to be used. On the other hand, there was a possi bility that the vanners in the concentrator had been run too flat, and therefore did not save all that could be saved. In this case a steeper setting of the vanners would increase the saving and would possibly make a re-treatment of the tailings unnecessary.

Since smooth belts only were in use, the first work was to determine the best adjustments of a Frue vanner with a smooth belt for treating fine feed. The second, to ascertain if a corrugated belt will do better on slime-feed than a smooth belt. Of course, any such difference in the saving for different adjustments, as reported above, could not be expected for the regular slime-feed, from which a large amount of concentrate can be easily extracted by almost any machine, while on a pulp which has already gone over a concentrating-table once, one machine may have hardly any effect, while another may save a considerable amount of mineral.

Some tests with different adjustments (mainly different slopes) convinced me that on this feed a higher slope also increased the saving. Since, however, this view was diametrical to the former practice in the concentrator of the Detroit Copper Co., accord-

ing to which slime-feed was treated on belts set as flat as possible (with the idea that the slimes should get a very small chance to run off), my views found opposition, and it was decided to have a contest, in which an experienced millman ran a vanner according to the established practice, while I ran another machine on the same feed according to my ideas. Two vanners were set side by side and fed by slime-pulp from a revolving distributor, so as to send the same amount of feed of the same quality to the two smooth-belt vanners. Four runs were made, averaging about 7 hr. each, from which the concentrates were collected, weighed, and assayed. One time the contestants changed machines between two runs. On an average of the four tests, the machine with the higher slope saved 10.1 per cent. more copper than the machine with lesser slope. One other point brought out by the contest was that, in order to effect a high saving, a certain amount of water, probably in excess of what is generally used on slime-vanners, is required.

The results of this contest indicated that it is possible to save a considerable part of the copper that heretofore has been lost, and that it would probably pay to determine accurately and systematically the most economical adjustments of a vanner in treating this feed.

The right way to find the best adjustments of a vanner for a certain feed is to vary all the elements that can be varied, and to determine the saving resulting from different combinations of adjustments. Since there are several variations possible, this scheme requires a large number of tests.

The principal elements which can be varied on the Frue vanner are: (1) the slope of the belt, (2) the amount of dressing-water, (3) the number of side-strokes, and (4) the speed of the belt. If each of these elements has a certain value, the machine will for a certain quality and quantity of feed produce a certain grade of concentrate. In practice, however, the grade of the concentrate which has to be made on the vanners will be a given quantity. It will be stipulated, for instance, to make a grade with only 10 or 20 per cent. of silica. This condition makes the above-mentioned adjustments interdependent. For instance, with a given number of side-strokes per minute, and a given slope of the belt, the vanner-man can

either use a certain amount of dressing-water and regulate the speed of the belt, or he can give the belt a certain speed and regulate the dressing-water so as to produce the desired grade of concentrate.

In most of these experiments the former way was chosen—namely, a certain quantity of dressing-water per minute was used. For this purpose a funnel was constructed, Fig. 1, having the opening, *A*, closed by a wooden plug, *B*, with an inserted glass tube, *C*. At the wide end of the funnel a tube, *D*, branches off. The funnel is set on a nipple, *E*, screwed on the pipe, *F*, which carries the dressing-water to the water-distributing box of the vanner. The water enters the funnel

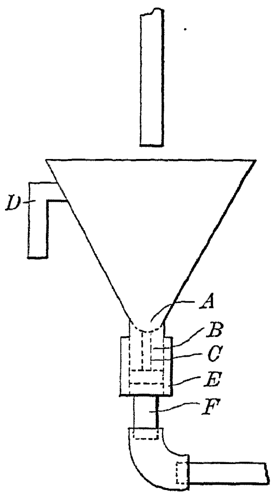


FIG. 1.—WATER-REGULATOR FOR VANNERS.

through its upper end. The inflow is so regulated that it fills the funnel and gives a small overflow through the pipe, *D*. As long as this small overflow is maintained the water running through the glass tube, *C*, is always under the same pressure, and the quantity passing through is constant. By changing the glass tube the quantity of dressing-water can be varied.

The quantity of feed was regulated in a similar way by passing the pulp through a short piece of iron pipe under a given head. In the later runs special efforts also were made to keep the consistency of the pulp uniform by using a separate settling-tank having a spigot-discharge controlled by a plug with an inserted iron tube, while the tank itself was supplied with the

same kind of pulp up to the settling-capacity for a clear-water overflow.

In order to get good results, it was found necessary to use a speed of the belt much more rapid than the ordinary adjustments allow. This was accomplished by replacing the cone-pulleys which regulate the belt-speed with larger ones.

The relative saving made on the two machines was determined by weighing, sampling, and assaying the concentrates produced in each run. Hand-samples of feed and tailings also were taken at regular intervals, but only the saving, based on weight and assay of the concentrates, was used for a comparison of the work of the machine under different conditions. In the present case this method of determining the saving is evidently the most reliable one. As a rule, the whole concentrate was dried and sampled so as to avoid errors due to faulty moisture-samples. At the start the concentrates were assayed by the cyanide method, but later these assays were used for preliminary work only, the calculations being based on electrolytic assays.

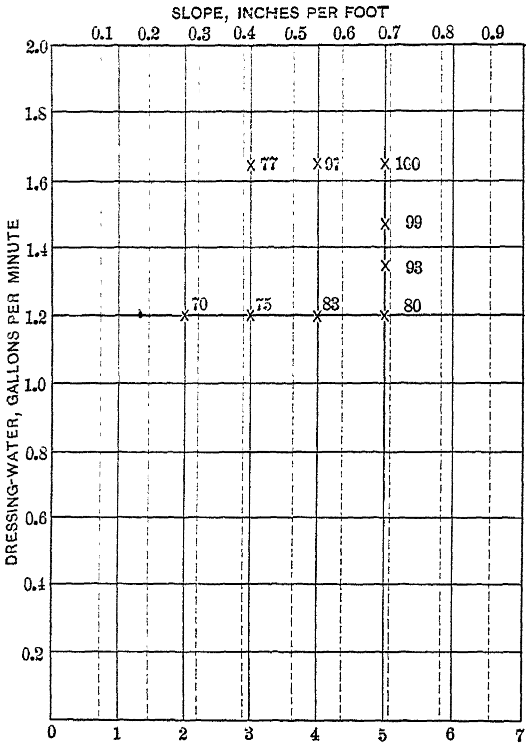
To eliminate errors due to the impossibility of keeping two machines in equally good condition all the time, the adjustments were reversed when practicable. For instance, when one machine had been run with a 3-in. slope and another with a 4-in. slope for one day, the next day the slopes were reversed, but since this called for frequent changes in both machines, which at times could not be easily effected, another method was generally followed. One machine, kept running in the same way for a whole series of runs, was called the standard vanner, and all the variations were made with the other machine. As an example, the results of one series of tests are given in Table I.

TABLE I.—*Effect of Variation of Slope.*

Test number		117 and 118	119	120	121	122	123	124	125
Slope.									
Between Posts.	In. per Ft.								
5 in.	0.680								
6 in.	0.816	....	.....	....	....	....	118.9	116.7	
7 in.	0.952	113.2	.....	....	....	114.2	....	....	114.8
8 in.	1.088	....	106	107.3	111.2				

The data in Table I. were obtained under the following conditions: dressing-water used, 2.7 gal. per min. (per 6-in. belt);

average quantity of ore treated per 24 hr., 8.95 tons; average amount of solid, 13.53 per cent.; average number of strokes per min., 188 (1-in. strokes); corrugated belt. The results are given in percentages of the results obtained by the standard vanner having a smooth belt, a 5-in. slope between posts (0.680



Smooth belt, 220 (1-in.) strokes per minute.
Feed: Lower vanner feed of D. C. M. Co. concentrator.
13.0 per cent. solid.
10.0 per cent. on 200-mesh screen (Denver Fire Clay Co.).
1.40 per cent. copper.
9.5 tons dry feed per 24 hours.
Saving is expressed in percentage of maximum saving obtained in this series of tests.

Fig. 2.—Variation of Saving with Dressing-Water and Slope.

in. per ft.), using 1.66 gal. of dressing-water per min., and operated at 229 (1-in.) strokes per minute.

Fig. 2, showing the results obtained on a vanner with a smooth belt, expresses the saving in percentages of the maxi-

mum saving obtained in this series of tests. The figures represent this saving, the slope and the dressing-water being used as co-ordinates. This diagram shows that, in order to make a good saving, it is necessary to set a vanner much steeper than is done in common practice. The best saving was obtained with 5-in. slope between the inside of the posts (0.680 in. per ft.), and $1\frac{5}{8}$ gal. of dressing-water per 6-in. belt. Under these conditions it was necessary to give the belt a speed of 120 in. per min., which is probably three times the rate of travel used in most mills, and far in excess of any speed that I have ever seen quoted. Both the large quantity of dressing-water and the high slope of the belt tend to produce a high speed. The results of these tests show that, in order to make a good saving on slime-feed with a vanner, it is necessary to give the belt a high speed, which can be done by using either a high slope or a large quantity of dressing-water. On a given feed the best combination can only be decided by experiment.

The details of Fig. 2 are not very complete, but since the determination of even these few figures took a long time, and since it was the expectation to discard the smooth belts should corrugated belts be found more economical, the data collected were considered sufficient.

Richards[1] mentions as extraordinarily high the rate at which the belts move forward in the Gates canvas-plant, operated at Jackson, Cal. At this plant very fine canvas-table concentrates are re-dressed on an end-shake vanner which has the extremely high slope of 1.5 in. per ft. I have tried repeatedly to use slopes approaching this one, but in every case with negative results, which may be due to the fact that the feed was not as fine or that an end-shake vanner will stand a higher slope. The amount of shaking-motion also is much larger in my tests, which may help to explain the difference. The shaking-motion is probably stronger than is practical in view of the difficulty of keeping the machines in good order, but as far as the saving is concerned, not much could be gained by reducing the strength of the motion, as will be seen later in this paper.

The question arises, why, if a high belt-speed gives so much better results, it is not used in some mills? Certainly, some

[1] *Ore-Dressing*, vol. ii., p. 660 (1903).

one must have tried high speed before, but I am inclined to think that no one ever investigated thoroughly the question of belt-speed. Most millmen trust a good deal to the eye, and a fast-traveling belt does not show the concentrate very plainly. Every one viewing two belts side by side, one at a fast speed and one at a slow speed, will feel certain that the slow-speed belt produces much more concentrate. The reason for this is, that in the case of the fast belt the concentrate is spread out over a larger surface in so thin a film that sometimes it is hardly perceptible. Other tests in the mills give misleading results. For instance, the belt which shows very little mineral behind the feed-box is frequently considered to make a good saving, but in some of our runs this test failed entirely. Belts showing mineral almost down to the tail-end often made much cleaner tailings and a better saving than did belts which looked clean over the entire length. Panning the tailings is more satisfactory; but the only safe way to determine the saving seems to be by actual assay.

One objection which has been raised against this way of running vanners is, that variation in the power would influence a fast-running belt more than a slow-running one. Whenever the driving-power loses speed a vanner begins to carry sand into the concentrates, on account of the gentler shaking-motion. And if a vanner be adjusted to the slower motion it will make too clean a concentrate as soon as the power recuperates, which means a loss of mineral, unless the vanner-man immediately re-adjusts the machine to the changed condition.

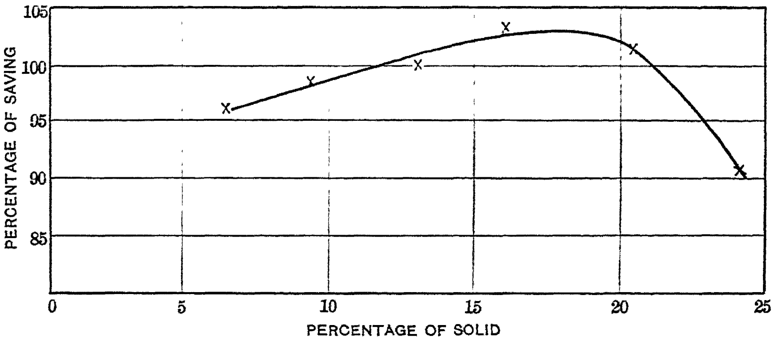
In other words, the losses will be heavier on a fast-traveling belt, on account of the higher slope. It has been found by experiment, however, that a machine with a high slope requires less regulation to meet changed conditions of power than a machine with low slope, and from this fact I infer that the losses due to lack of attention to the changes in power will be reduced, and not increased, by giving the vanners a high belt-speed. This condition, therefore, recommends a high belt-speed, particularly in places where the power is not very constant.

It may also be mentioned that a violent shaking-motion makes a vanner more independent of changes in the power, so that for plants having poor power it is advisable to use

a rapid belt-speed and shaking-motion. While high speed will cause a belt to wear out faster than a slow speed, the results obtained at the concentrator of the Detroit Copper Co. show that the cost of increased wear will be made up many times by the increased saving.

The feed treated on the vanners in the tests shown in Fig. 2 averaged 13 per cent. of solids. Screen-analyses made at different times showed between 6 and 14 per cent. of residue on a 200-mesh screen of the Denver Fire Clay Co. The average load was 9.5 tons (dry) per 24 hours.

We also investigated the consistency of pulp best suited for



Smooth belt, 209 (1-in.) strokes per minute, 2.20 gal. dressing-water per minute 4-in. slope between posts (0.544 in. per foot).
Feed: Lower vanner feed of D. C. M. Co. concentrator.
9 per cent. on 200-mesh screen (Denver Fire Clay Co.).
1.48 per cent. copper in feed.
7.93 tons dry feed per 24 hours.
Saving is expressed in percentage of saving made with a pulp of 12 per cent. solids.

FIG. 3.—VARIATION OF SAVING WITH PERCENTAGE OF SOLIDS IN FEED.

treatment on vanners. The experiments made were similar to those described above, except that the feed was thickened in a settling-tank with clear-water overflow before it reached the distributor. Alternating daily, extra water was applied to one of the two machines, and in this manner the saving determined which corresponded to different percentages of solid matter in the pulp. The curve in Fig. 3 shows the saving obtained under different conditions, and proves that thickening the pulp beyond a certain limit decreases quite rapidly the economy of a vanner. It should be observed that in these experiments the settling of the pulp has been carried much further than

would be done in practice. A pulp of this material containing 24 per cent. of solid has almost the consistency of syrup. To convey an idea of the thickness of the feed, the curve, Fig. 4, has been drawn, showing the settling in a cylinder 22.25 in. high. It proves that it takes a long time to settle pulp to this consistency; consequently, if this were carried out in practice it would require a very large settling-capacity.

The adjustments of the vanners in these experiments having been kept the same, the question arises, is it right to assume that the adjustments found the best in the former tests will also be the best for thickened feed? In other words, may not a very

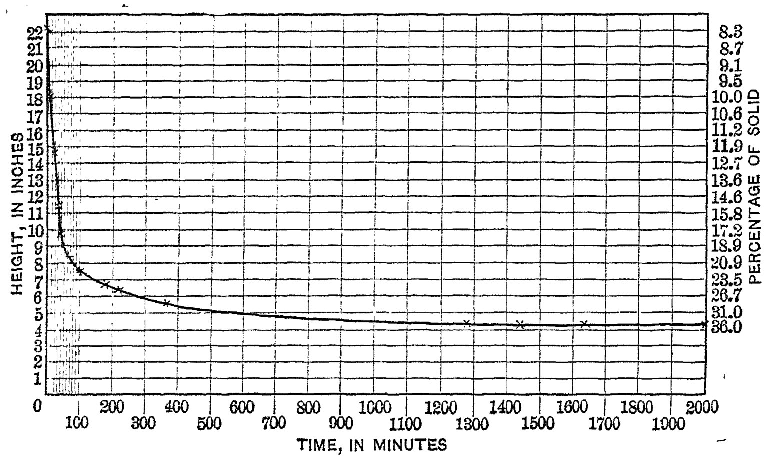
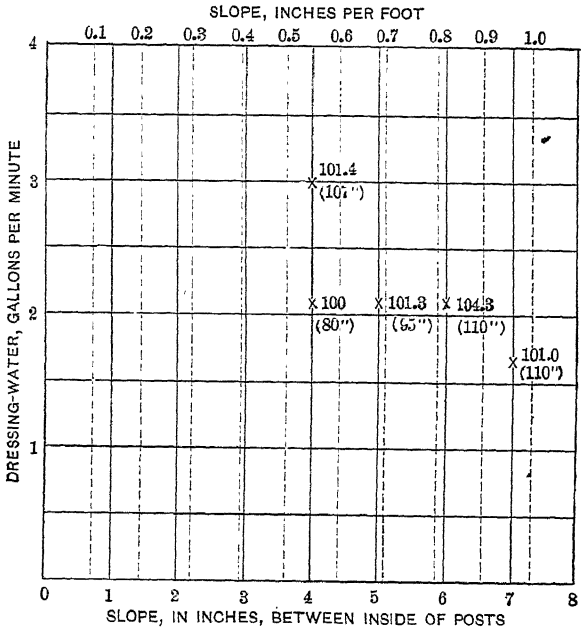


FIG. 4.—SETTLING OF LOWER VANNER FEED (D. C. M. CO. CONCENTRATOR) IN 22.25-IN. CYLINDER.

thick feed require a different adjustment? This question was investigated by the tests represented in Fig. 5, with a pulp averaging 25.5 per cent. of solids. At first, two chances for improvement seemed to exist: one, the use of a higher slope, in order, as far as possible, to spread out the thick pulp and effect a greater contact with the belt; the other, the use of much dressing-water in order to counteract the deficiency of water in the pulp. The outcome of the experiments showed that spreading out the pulp by increasing the slope does not counteract the harmful effects of the deficiency of water in the pulp, but it also showed that an increase of the quantity of dressing-water off-sets the disadvantages of a too-thick pulp.

Since the data for the saving in Fig. 5 are expressed by the saving of another vanner with equally thick pulp, the high saving of 107.4 per cent. means only so much more saving than can be made on a vanner running with 4-in. slope (0.544 in. per ft.) and 2.2 gal. of dressing-water treating feed of the same thickness. However, the results show that an extremely thick



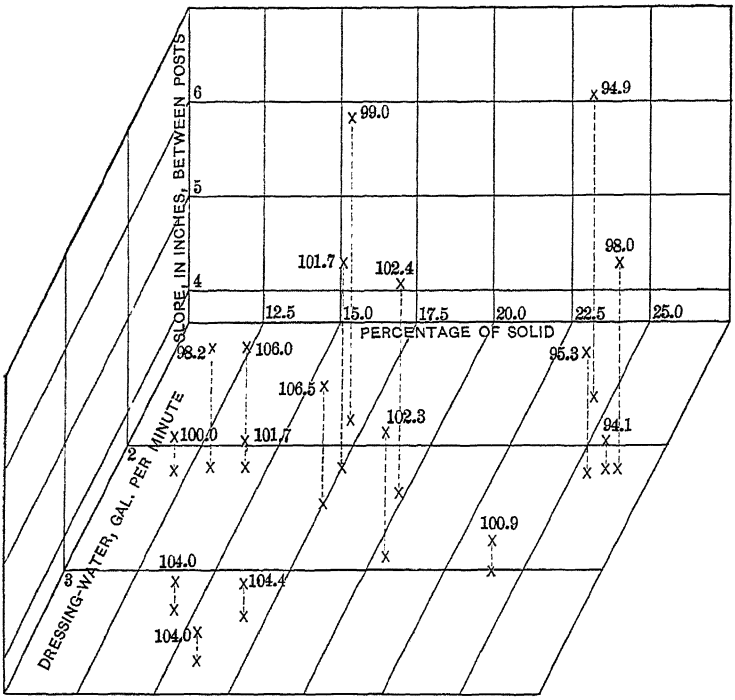
Smooth belt, 210 (1-in.) strokes per minute.
Feed: Lower vanner feed of D. C. M. Co. concentrator.
25.5 per cent. solid (average).
9 per cent. on 200-mesh screen (Denver Fire Clay Co.).
1.46 per cent. copper.
6.2 tons dry feed per 24 hours.
Saving is expressed in percentage of saving made with 4-in. slope between posts (0.544 in. per foot) and 2.2 gal. dressing-water per minute.
Figures in parentheses mean travel of belt in inches per minute.

FIG. 5.—VARIATION OF SAVING WITH DRESSING-WATER AND SLOPE FOR VERY THICK FEED.

feed can be treated fairly well with a large quantity of dressing-water. The tests, Figs. 4 and 5, are also shown in Fig. 6, together with the results of other test-runs. It seems from Fig. 6 that the best results can be obtained with a slope of 5 in. and a pulp-thickness of about 16 per cent. of solid.

Fig. 6, the saving effected, is represented in its relation to

the three variable quantities which determine it—namely, the percentage of solids in the feed, the slope of belt, and the amount of dressing-water. The saving made by each combination tested is expressed by the figure attached to the point representing this combination. The distance of each of these points from the base-plane, indicated by the length of the per-



Smooth belt, 210 (1-in.) strokes per minute.
Feed: Lower vanner feed of D. C. M. Co. concentrator.
9 per cent. on 200-mesh screen (Denver Fire Clay Co.).
1.47 per cent. copper in feed.
7.10 tons dry feed per 24 hours.
Saving is expressed in percentage of saving made with 4-in. slope between posts (0.544 in. per foot) and 2.2 gal. dressing-water per minute.

FIG. 6.—TESTS WITH THICKENED FEED.

pendiculars (the dotted lines), expresses the slope; the position of the foot-point of these perpendiculars in the base-plane shows the corresponding figures for the quantity of dressing-water and for the percentage of solid in the feed.

These results, so far as the percentage of water in the feed is concerned, cannot be applied to other ores, since the consist-

ency best suited for treatment on concentrating-machines depends largely on the composition of the ore (the percentage of clay, etc.). I think, however, that a pulp of the same degree of fineness which will settle with the same velocity as the samples treated would give nearly the same results; also, that results for pulps of equal settling-velocity would yield a better comparison than pulps of equal percentage of solids.

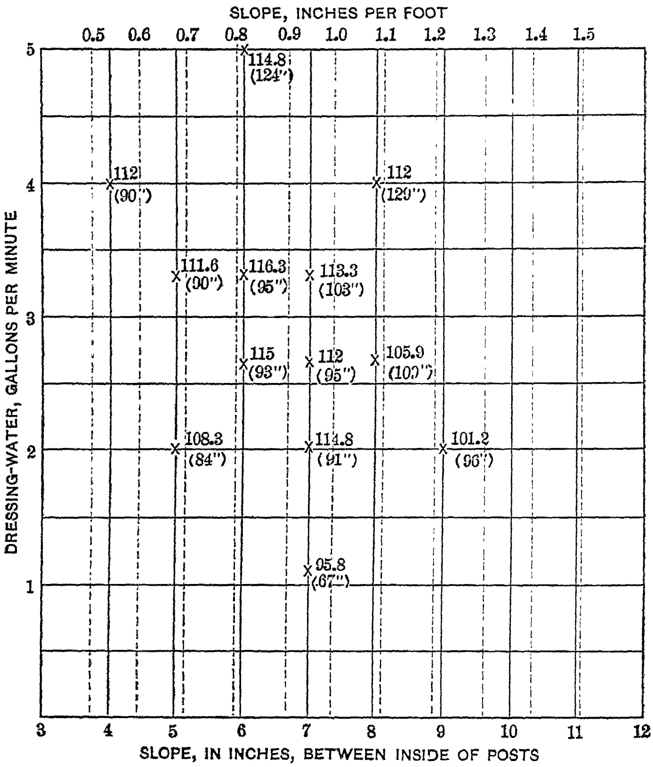
Other tests were made to determine the best adjustments for a corrugated belt treating this slime-feed in the same way in which the smooth belt had been tested before. Since these experiments were carried out before the preceding series, no use was made of the fact that the saving could be improved by slightly thickening the pulp. Otherwise, it might have been more logical to conduct these experiments with the corrugated belt with a somewhat thickened feed. The results of these tests are represented in Fig. 7, in which the saving is expressed in percentages of the saving made on the vanner with the smooth belt, which was taken as the standard. The corrugated belt used was an old one, the corrugations of which were much worn and rounded by long use.

From the results given in Fig. 7, it follows that the best slope for this belt is 6 in. between the posts, or 0.816 in. per ft. The belt-speed which had to be used in each run is added in parentheses to the figures representing the saving. The speed necessary for a good saving is not quite as high as that of the smooth belt, probably on account of the lower number of side-strokes used. The figures show also that a far larger amount of dressing-water was necessary than for the tests with the smooth belt.

All the tests had shown that a high belt-speed is most advantageous in saving slime. Experiments were then tried to ascertain the possibility of increasing the belt-speed, and accordingly the saving, without giving the belt, as a whole, too much fall or applying too much dressing-water, simply by raising the front part only of the belt. The higher fall of the front end was gained by raising the front roller about 1 in. and the second roller enough barely to support the belt. The resulting increase in the rate of belt-speed amounted to about 10 in. per min. In every case, however, the effect on the saving was less than 1 per cent. In one case there was a small gain, in two cases a small loss, so that it seems safe to conclude that the effect of

this change is very small and hardly larger than the errors connected with these tests.

Tests were made to determine the influence of the shaking-motion on the saving made by a vanner. Some attempts had

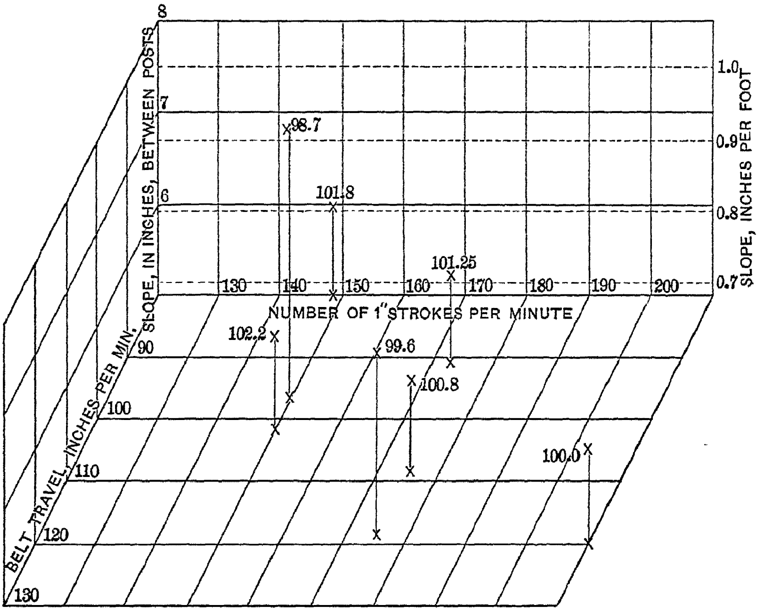


Corrugated belt, 190 (1-in.) strokes per minute.
Feed: Lower vanner feed of D. C. M. Co. concentrator.
12.64 per cent. solids.
9.0 per cent. on 200-mesh screen (Denver Fire Clay Co.).
1.4 per cent. copper.
8.43 tons dry feed per 24 hours.
Saving is expressed in percentage of saving made on standard vanner with smooth belt, 229 (1-in.) strokes per minute, 5-in. slope between posts, 123 in. belt-travel per minute.
Figures in parentheses mean travel of belt in inches per minute.

FIG. 7.—VARIATION OF SAVING WITH DRESSING-WATER AND SLOPE.

been made before to gain information on this question. But at that time the power for the experimental machines was quite variable, and the only result obtained was that any considerable reduction in the shaking-motion was practically impossible,

because the machines were influenced too much by the changes in the power. Later, better power was secured, and although variations of 2 per cent. to either side in the number of strokes still occurred frequently, it seemed to be improvement enough to consider a reduction in the speed of the shaking-motion. The results, Fig. 8, do not show a great improvement to be



Corrugated belt (D. C. form).
Feed: Lower vanner feed of D. C. M. Co. concentrator.
14.4 per cent. solids.
10.0 per cent. on 200-mesh screen (Denver Fire Clay Co.).
1.44 per cent. copper.
8.80 tons dry feed per 24 hours.
Saving is expressed in percentage of saving made on vanner with corrugated belt (D. C. form), 210 (1-in.) strokes per minute, 6-in. slope between posts (0.816 in. per foot), 100 in. belt-travel per minute.

FIG. 8.—VARIATION OF SAVING WITH NUMBER OF STROKES.

gained by reducing the speed, but indicate that a reduction of the number of side-strokes to about 150, with a considerable increase in the amount of dressing-water in order to gain high belt-speed, would be an improvement. A reduced side-motion requires more dressing-water to produce the same belt-speed.

Very probably such an extensive reduction would prove impracticable with our power-conditions, as one man has to operate

a large number of machines; but doubtless some reduction would be beneficial, especially with our high rate of shaking-motion, which, in spite of continuous attention, shakes the machines loose very soon, and this, of course, means loss.

As already mentioned, Fig. 7 shows the saving made by the corrugated belt in percentages of the saving made by the smooth belt. These runs were originally not intended to determine which belt made the best saving, but only to find the best setting for the corrugated belt, using the smooth belt as a standard in all these tests. The saving made on the corrugated belt was, however, in all these tests so much higher that it seemed safe to conclude that the corrugated belt was better than the smooth belt for the treatment of slime-feed. This result was doubted by many experienced in the concentration of copper-ores, and a second contest was arranged in which I operated the corrugated belt against several experienced concentrating-men using a smooth belt. The smooth belt was nearly new, having served in the concentrator for some months. The corrugated belt was the one used in the former tests, and before this had been in constant use for a long time in another mill, so that it was not in first-class condition. In some places the riffles had been worn away almost completely, while in others they still stood out prominently. Besides, there were numerous bad places, due to rough handling in shipping and repeated putting on and taking off the belt. At the bottom of the riffles the canvas was exposed.

In the first test of the new series the slope of the smooth belt was the same as before, 2 in. (0.272 in. per ft.). The results obtained were:

Lower Vanner Feed.—10 per cent. on 200-mesh screen; 12.31 per cent. of solid matter; 1.30 per cent. of copper; 8.5 tons (dry) per 24 hr. Running-time, 6 hr. 40 min.

	Smooth Belt.	Corrugated Belt.
Slope between posts, inches,	2	6
Slope per foot, inches,	0.272	0.816
Belt-travel per minute, inches,	60	120
Strokes per minute,	234	194
Concentrates, dry weight, pounds,	82.63	160.90
Concentrates, Cu, per cent.,	20.66	17.39
Copper saved, pounds,	17.07	27.98
Tailings-assay, Cu, per cent.,	0.97	0.79

Saving of corrugated belt in percentage of smooth belt, 163.9 per cent.

In the following tests no limitations were set to the adjustments; 12 days were devoted to these tests, including work with the table mentioned before, which received feed from the same feed-distributor. The average results were:

Lower Vanner Feed.—8.81 per cent. on 200-mesh screen; 11.91 per cent. of solid matter; 1.49 per cent. copper; 8.09 tons (dry) per 24 hr.; running-time, 83 hr. 30 min.

	Smooth Belt.	Corrugated Belt.
Slope between posts, inches,	3.96	6.00
Slope per foot, inches,	0.538	0.816
Belt-travel per minute, inches,	100	116
Strokes per minute,	204	195
Concentrates, dry weight, pounds,	1,438.6	1,622.2
Copper saved, pounds,	266.23	300.35
Concentrates, Cu, per cent.,	18.51	18.51
Tailings-assay, Cu, per cent.,	0.824	0.797

Saving of corrugated belt in percentage of smooth belt, 112.8.

For a further comparison between the work of the corrugated and the smooth belt, a test was made using as feed the tailings from the slime-vanners (smooth belts) in the concentrator. The results of this test were:

Lower Vanner Feed.—Tailings from lower vanners, running with smooth belts, about 3.5 in. slope between posts (0.476 in. per ft.); 9 per cent. on 200-mesh screen; 10.73 per cent. of solids; 0.77 per cent. of copper; 6.90 tons per 24 hr.; running-time, 10 hr. 30 min.

	Smooth Belt.	Corrugated Belt.
Slope between posts, inches,	4	6
Slope per foot, inches,	0.544	0.816
Belt-travel per minute, inches,	95	98
Strokes per minute,	207	207
Concentrates, dry weight, pounds,	54.5	82.56
Concentrates, Cu, per cent.,	12.09	10.46
Copper saved, pounds,	6.59	8.64
Tailings-assay, Cu, per cent.,	0.68	0.67

Saving of corrugated belt in percentage of smooth belt, 131.7.

This result confirmed the experience gained in the first tests, that the saving effected by the corrugated belt on these slime-tailings far exceeds the saving made by the smooth belt.

The tests with the corrugated belt had been made using the old belt, the corrugations of which were greatly worn down. Since the results were so very encouraging, a new belt of the same pattern and having sharp corrugations was secured and tested. The saving effected was extremely disappointing. It

was not possible to get even as good results as with a smooth belt. Later, a large number of belts were made with exactly the same shape of riffles as had been formed by the wear.

With one of these belts tests were run against both the smooth and the old corrugated belt, which showed that the new belt, after it had been on the machines for a couple of weeks, was just as good a saver of mineral as the old one, and was considerably superior to the smooth belt. One difference existed—namely, that the old belt, probably on account of its roughness and of the exposure of the canvas at the bottom of the riffles, carried some very fine slimes into the concentrate, which, however, was of rather low grade, and did not influence the saving very much.

For a test on a larger scale, 10 machines, equipped with the new corrugated belts, were run against 10 machines with smooth belts. The concentrate from each group of machines was caught in a wooden tank large enough to hold all the concentrate produced in the course of a week. Five tests were made, the first one lasting seven days, the others five days each. The results of the first test (which favored the corrugated belt) were not accepted, since there was some doubt of the correctness of the work. In the fourth and fifth tests, the feed-pipes which had been supplying the feed for the smooth belts in the first and second tests supplied the corrugated belts, and *vice versâ*, in order to eliminate possible inaccuracies of the distributor. The smooth belts used for the tests had been running in the concentrator for some time. The rubber was worn off from the back surface of some of the belts and the canvas exposed. To prevent the removal of mineral from the vanner-box on this rough surface, spray-water was used to keep the surface clean.

The average result of these tests, occupying 20 days altogether, was that the corrugated belts produced 123.2 per cent. of the copper produced on the smooth belts. The grade of concentrate on the corrugated belts was 20.04 per cent. of copper, as compared with 19.39 per cent. on the smooth belts.

The corrugated belts were set with 4.92 in. slope (0.669 in. per ft.) and the smooth belts with 3.56 in. (0.484 in. per ft.), which is a little less than the slopes found best in the experiments. But the ordinary load in the mill is smaller than the

load used on the experimental machines, so that a gentler slope may be justified. The results obtained are decisive enough to permit the statement that the corrugated belt of the form described above has proved a better slime-saver than a smooth belt.

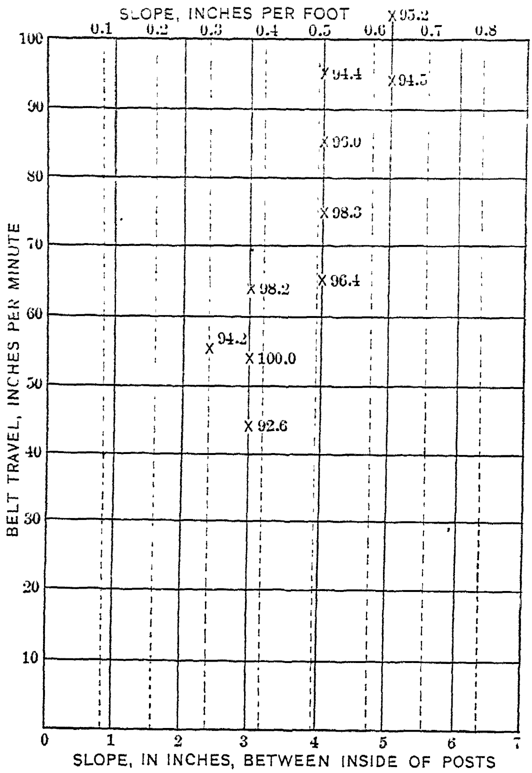
It is often pointed out as an objection to a corrugated belt that it will make a dirtier concentrate, although it may make a better saving. Expressed so broadly, this statement is certainly incorrect. Since sand cannot be noticed as easily as on a smooth belt, it requires practice to produce a uniformly clean concentrate on a corrugated belt. But there is not the least doubt that, if the concentrate made on a given machine is not clean enough, it can be raised to any degree of purity by the ordinary adjustments of vanners. My tests show that the new corrugated belt produces a larger amount of concentrate of the same grade than does a smooth belt.

Calculations based on the result of the test determining the saving that could be made by re-treating the tailings from the slime-vanners on corrugated belts had shown that it would hardly pay to install additional machines for this purpose with copper at a price of 13 cents per pound. Replacing the smooth belts in the concentrator by corrugated belts will improve the saving of the vanners and make the re-treatment of tailings from the slime-vanners decidedly uneconomical.

Possibly it may be profitable to classify the tails thoroughly before attempting re-treatment, but since these slimes contain only a small percentage of material that will stay on a 200-mesh screen, it is not probable that this suggestion will lead to any improvement. Besides, if classification yields a better saving, it would be better to provide a thorough classification for the vanner-feed.

Another way of raising the saving of the slime-vanners is to lower the tonnage treated per machine, and with this end in view experiments were undertaken to establish the relation between the saving made and the load carried on these vanners. The feed-distributor was provided for these tests with compartments of unequal size, so that more feed could be sent to one vanner than to the other one. The proportion in which the feed was distributed was carefully determined. The results of these tests are represented graphically in Fig. 10, which shows

that within the limits tested the vanner carrying the lighter load makes the better saving. The improvement resulting from reducing the load is not very great. For the present grade of milling ore, with copper at 12.5 cents, and with the present cost



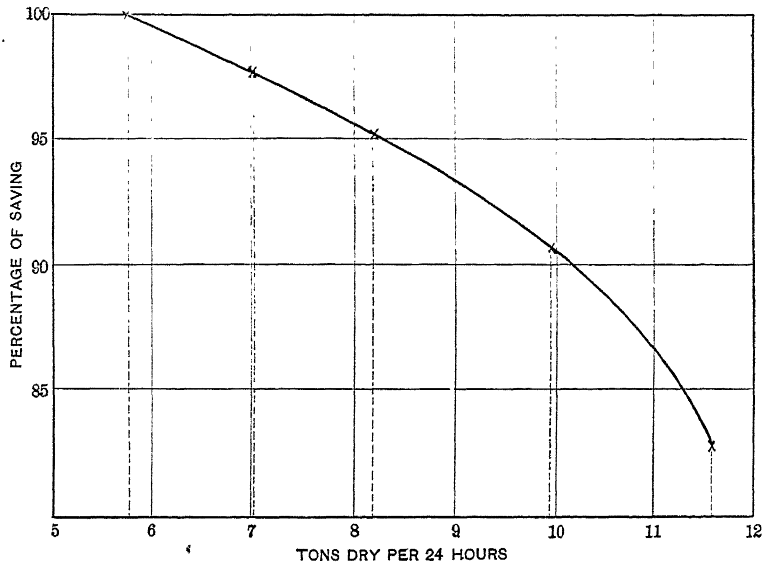
Corrugated belt, 210 (1-in.) strokes per minute.
Feed: Upper vanner feed East, D. C. M. Co. concentrator.
12.65 per cent. solids.
27.2 per cent. on 100-mesh screen, 43.6 per cent. (cumulative) on 200-mesh screen (Denver Fire Clay Co.).
1.51 per cent. copper.
6.74 tons dry per 24 hours.
Saving is expressed in percentage of saving made on corrugated-belt vanner with 210 (1-in.) strokes per minute, 3-in. slope between posts (0.408 in. per foot), 54 in. belt-travel per minute.

FIG. 9.—TESTS WITH COARSER FEED.

of labor, power, repairs, etc., the most economical load for the new corrugated belts is between 8 and 9 tons, which shows that under existing conditions a reduction of the present load is not advisable. For higher copper-prices, however, a smaller load

would be more economical. At 20 cents the greatest economy is obtained with a load of about 7 tons.

The slope which is to be given to corrugated belts depends very largely on the kind of material treated. Formerly it was the rule at the Detroit concentrator, and I suppose many others, to treat the finest feed with the lowest, and the coarsest feed with the highest, slope. That the contrary is rational has been



Corrugated belt (D. C. form), 210 (1-in.) strokes per minute, 6-in. slope between posts (0.816 in. per foot), 120 in. belt-travel per minute.
Feed : Lower vanner feed of D. C. M. Co. concentrator.
14.7 per cent. solids.
10.0 per cent. on 200-mesh screen (Denver Fire Clay Co.).
1.31 per cent. copper.
Saving is expressed in percentage of saving made on a vanner with the same adjustments carrying a load of 5.77 tons dry per 24 hours.

FIG. 10.—VARIATION OF SAVING WITH LOAD.

held by Richards and others. The experiments, graphically represented by Fig. 9, show that on a coarse feed the higher slope gives poorer savings than a lower slope. The fact that a feed not very different requires such a different treatment, seems to point to close classification as an improvement for vanner-work, and to proper adjustments of each machine to the pulp treated.

This question is still undecided, but I hope that some light

has been thrown on a few points by the work described in this paper. If nothing else, it shows that it pays to study questions of this character.

NOTE (July, 1909).—Since writing this paper some additional tests have been made, in which the corrugated belts showed no marked superiority over smooth belts. These tests were made under mill conditions, 10 corrugated-belt machines running against 10 others with selected smooth belts, both sets of machines operated with high belt-speed. The feed was somewhat coarser than in the tests referred to in the paper. Although, therefore, the results are not exactly comparable, they could be interpreted as throwing some doubt on the correctness of the previous tests investigating this point. For this reason I have intended for a long time to repeat these tests. But as some changes in the operation of the mill have made it impossible so far to obtain the same kind of feed as used in the former tests, and as there may not be an opportunity of carrying out this plan for some time, I thought it better not to delay the publication any longer. I felt, however, that I should call attention to this apparent discrepancy.

Biographical Notice of Hermann Wedding.*

BY EMIL SCHROEDTER, DÜSSELDORF, GERMANY.

(New Haven Meeting, February, 1909.)

THE death, on May 6, 1908, of Dr. Hermann Wedding, Privy Mining Councilor of the Kingdom of Prussia, and Professor of the Metallurgy of Iron and Steel at the Royal Mining Academy of Berlin, was a loss keenly felt by the whole metallurgical world. Not alone in his native land, but also in foreign countries, did he enjoy an esteem of which the title of honorary membership in the American Institute of Mining Engineers, the United States Association of Charcoal Iron-Workers, and the (British) Iron and Steel Institute bears eloquent testimony.

---

* Translated by the Secretary, and accompanied with portrait furnished by the courtesy of *Stahl und Eisen*, constituting the frontispiece of this volume.

Hermann Wedding was born March 9, 1834, in Berlin. At the suggestion of Dr. C. J. B. Karsten, the veteran father of Prussian metallurgy and author of the first German text-book of the metallurgy of iron, he turned his attention, after graduating from the gymnasium, to mining and metallurgy. In 1853, he began a practical course at the Malapane iron-works, in Upper Silesia. Subsequently, he studied at the Berlin Mining Academy, and received from the University in 1859 the degree of Doctor of Philosophy. A journey which he made to England, somewhat later, led to his undertaking a German translation, with revision and additions, of John Percy's classic treatise on the metallurgy of iron and steel, the publication of which, in successive installments, stretched through many years.

In 1863, he published, as a thesis in connection with his "Assessor's examination," an essay, entitled The Results of the Bessemer Process for the Production of Steel, and Its Prospects for the Rhenish and Westphalian Iron and Steel Industry, which revealed such ability and knowledge as to secure for him a call to the chair of metallurgy in the Berlin Mining Academy.

He began his lectures in 1863, and this position and work, more congenial to his taste and talent than any other, afforded him abundant satisfaction to the end of his life. At first, he dealt with general metallurgy and assaying, but soon extended his activity as a teacher into all metallurgical departments, becoming at last specially useful and famous in that of iron and steel. In his long and fruitful career he understood how to keep himself always in touch with practice. Moreover, he regarded it as his duty to maintain relations with his pupils outside of the class-room. In connection with the frequent journeys of observation which he made with them, he was unwearied in endeavors for their benefit, explaining to them every day in thorough, helpful fashion all that they had seen, and joining heartily in their social evening merriment.

As an author he was exceptionally fertile. In the first rank of his works must be placed his *Ausführliches Handbuch der Eisenhüttenkunde*, which made him known in all civilized countries. The first edition of this monumental work appeared in three parts, in 1864, 1868, and 1874. Two supplementary volumes followed, in 1884 and 1887. Although, as re-

marked above, the work appeared as a German version of that of Percy, it contained much new material, and, especially in its later portions, often departed from the English original. The second edition deserves to be regarded as an entirely new work, though under the old title. Of this edition, the final section (the second part of the fourth volume) was left unfinished. Wedding was destined not to complete this greatest work of his life, which had always lain nearest to his heart, and upon which he labored with intense zeal in his last years.

Aside from this, his contributions to technical literature, through *Stahl und Eisen*, the *Transactions of the Verein zur Beförderung des Gewerbefleisses*, and other journals, foreign as well as German, were exceedingly numerous. In the *Transactions of the American Institute of Mining Engineers* he published the following:

Title.	Volume.	Page.
Remarks on the Hot Blast,	V.	79
Remarks on the Expulsion of Cinder in Rolling Rails,	V.	115
The Nomenclature of Iron,	V.	309
The Progress of German Practice in the Metallurgy of Iron and Steel Since 1876, with Special Reference to the Basic Processes,	XIX.	331
Remarks on Microscopic Metallography,	XXII.	246
Remarks on the Open-Hearth Process,	XXII.	691
A Biographical Notice of Oberberghauptmann Dr. Albert L. Serlo,	XXIX.	99

Through his two visits to the United States, in 1876 and 1890, and his participation in the receptions given to members of this Institute in Germany in 1888 and 1906, Councilor Wedding became personally well known to many of his American colleagues as a welcome and congenial companion, whose alert and eager mind was united with a sympathetic and humorous temperament.

He was twice married, and had eight children by his first wife and five by his second. Ten survive him. His eldest son is captain of a corvette in the Imperial German navy; his second son, a Councilor in the Imperial Foreign Office at Berlin.

Death overtook him on May 3, 1908, while he was taking part at Düsseldorf in the general meeting of the German Society of Ironmasters. Although he had been for some time feeling unwell, he attended with characteristic punctuality and

sense of duty all the sessions and addresses, until he was prostrated by illness, which soon showed itself to be an apoplectic stroke, terminating after three days in death. Such a quick departure he had always desired, and those who were with him in his last hours will never forget the happy smile which still lingered on his lips, already shadowed by the wing of the Angel of Death.

Iron endurance, never-failing power to work, clearness of perception, an unmistakable gift for statement, explanation, and instruction, and loyal fidelity to duty—these were the constant characteristics of Wedding's long, active, useful and honorable life to its last conscious moment. By virtue of these qualities he won his high place in the esteem of both students of the science and practitioners of the art of the metallurgy of iron and steel. Only a mind so quick, an industry so ceaseless, and a devotion so absolute, could have felt and followed the irresistible attraction presented by innumerable new problems and developments, which have arisen in unbroken succession, from the day of Wedding's first "shift" of practical service at Malapane, in 1853, to the present time. That period has seen the multiplication of our German production of iron to fifty-fold its magnitude in 1853, as the result of repeated revolutionary improvements in our metallurgical works, and of discoveries in the chemical, mechanical, and electrical fields, which have transformed the field of our profession, not only in the past, but for a progressive future, not yet revealed.

Wedding died "in harness,"[1] as he wished to die. And he may well have said with pride, and gratitude, what we can now say for him, that it was his great good fortune to begin his work at the beginning of a wonderful era of progress in the German iron industry, and that with keen insight and marvel-

[1] SECRETARY'S NOTE.—The German phrase employed by Dr. Schroedter, "Er starb in den Sielen," is an East-Prussian farmers' phrase, referring to a faithful horse that dies with the harness on him, in the midst of his work. It was Bismarck who first introduced it into German literature, in which it is now universally recognized as a felicitous figure. Curiously enough, I have been able to translate it literally, by an equally current phrase which carries a different simile. For "harness," in our figure, means the armor of a knight, and "dying in harness" means dying in battle. And, still more curiously, either figure is equally applicable here, if we but recognize that the soldiers who contribute to the victories of peace are the knights of a new and higher chivalry.—R. W. R.

ous enthusiastic industry he had kept pace for more than half a century with all departments of this progress, contributing to it, as chronicler, critic, and teacher, an indispensable element.

The exact value of his contribution it may be left for history, with stronger light and longer perspective, to measure. But it seems safe to say now, that he was probably the last of those who could fairly claim to command the whole of this field, as Humboldt and Leibnitz, in their generations, commanded respectively the fields of physical science and of mental philosophy.

Postscript.

By R. W. Raymond, New York, N. Y.

After translating, with much pleasure, the foregoing brief, but discriminating and just, as well as sympathetic, notice by Dr. Schroedter, I add, with his permission, a few remarks to be published with it.

In the first place, I would express in behalf of the many American members of the Institute who knew Professor Wedding, our hearty sense of his winning and commanding personal character, as well as his professional eminence, and our sincere grief at his death, which means to us the departure of a genial and loyal friend.

In the second place, I wish to call attention with renewed emphasis to the nature of his work, as already intimated in the above notice. For it seems to me that the labors of the historian and critic of technical progress are too generally underrated, and that the man who sacrifices the opportunity of winning fortune and fame by new inventions, in order to accept the less conspicuous yet more important service of consolidating and recording the results achieved by inventors, and handing them down to future generations as the indispensable basis of future progress, does not always receive the reward which he deserves. I do not remember that the name of Wedding is borne by any brilliant, original discovery, theory or process. Yet he who furnishes only a happy thought may be less meritorious than he who patiently devotes a long and unselfish life to recording, arranging, explaining, and rendering truly fruitful the happy thoughts of others. This function, indeed, is the more difficult, and, I think, after all, the more useful one.

Modern Progress in Mining and Metallurgy in the Western United States.

PRESIDENTIAL ADDRESS.

BY DAVID W. BRUNTON, DENVER, COLO.

(Spokane Meeting, September, 1909.)

I. INTRODUCTION.

THE list of our past-Presidents comprises the names of many who, in their official addresses, have sketched the current progress of the arts and professions with which they were familiar. Such addresses are, in my judgment, highly useful, setting forth the results of our own work as a society, and recalling to our minds the particular lines in which discussion would prove most fruitful. Following this course with some hesitation, I venture to offer an outline of the recent improvements and the present situation in mining and metallurgy in the Western United States.

The wonderful advances in mechanics, chemistry, and electricity have all combined to aid these arts to such an extent that progress during the last decade has undeniably been more rapid than ever before in the history of the profession. In all mining countries improved transportation-facilities, in many instances called into existence by the traffic created by the mines themselves, have done much to enlarge the field of operations by the reduction of freight-, operating- and living-expenses, thereby bringing lower-grade properties into the producing class. Under normal conditions, as the age of a district increases, all these different factors should combine to off-set the augmented costs attendant on deep mining, and greatly tend to prolong the profitable life of the mines.

The more recent and important improvements in our Western mining-practice which have contributed most towards the advancement of the art may be briefly summarized as follows; but, before beginning this portion of my address, I wish to thank most heartily the many engineering friends who have so

kindly furnished data covering their latest practice, without which this paper would have been even less complete than in its present form.

II. Mine-Mapping.

Not many years ago most mining companies thought it amply sufficient to have a surface-map of their properties and a composite map showing the different underground workings in their mines. To-day, almost every important concern maintains, in addition to the above, both stope- and assay-maps, while many of the larger companies add individual-level horizontal and vertical cross-section maps showing the underground geology in full. Upon these maps conventional designs in black ink are used to designate the various rocks, while the different veins or vein-systems are shown in colors. These sections are frequently drawn also upon glass sheets, which are then inserted in wooden frames provided with vertical or horizontal slots or grooves cut at the proper relative distances apart to correspond with any desired planes of cross-section or with the working-levels of the mines in question. The great advantages of such plans and sections cannot be overestimated. They not only show at a glance the tonnage and value of the ore in sight, but also afford a guide for development-work, whereas the old-fashioned maps were nothing but a record of the work performed, and were practically useless for any other purpose. The improvement named has brought about another, the importance of which is just beginning to be recognized. I refer to the employment by large mining companies of economic geologists, who are not burdened with the duties of surveying, directing workmen, etc., but give their whole attention to the geological problems encountered in the work. The advice of such experts in the purchase of property, the running of exploration-drifts, the location of shafts, etc., and the interpretation of local fault-systems, and other structural features, has already proved of inestimable value to their employers.

III. Surface-Mining.

Large ore-bodies occurring near the surface can, in many cases, be most cheaply and satisfactorily mined by stripping off the overburden and loading the mineral into cars by either the

"milling" or the steam-shovel system. In the West, steam-shovel mining is confined almost entirely to the low-grade copper-deposits at Ely, Nev., and Bingham, Utah. The system employed follows closely that of many iron-mines of Minnesota; and 95-ton shovels, with 3.5-cu. yd. (7-ton) dippers, are in common use. At Bingham, the Boston Consolidated Co. stripped the overburden from its deposit at the rate of 200,000 tons per month. The maximum amount handled in a calendar month was 282,903 tons, in August, 1907, and the maximum tonnage for a single day, with four shovels, is 15,000 tons. The Utah Copper Co., immediately adjoining this, is also carrying on equally extensive stripping and mining; and as many as 13 steam-shovels and 26 locomotives have been counted at work within a radius of half a mile.

IV. Rock-Drills.

The great improvements in core-drills, both diamond and calyx, enable us to-day to explore ground hundreds of feet in advance of the actual openings and afford great aid in all development-work. Power-drilling has now almost entirely replaced hand-work, and a vast assortment of drills has been placed on the market, from which a careful engineer will often have extreme difficulty in selecting the machine best adapted for a particular service. Rock-drills, reciprocating, air-hammer, and electric-air, are all in successful operation to-day; and with the steady improvement, both in design and material employed in construction, there is every reason to believe that the drill of the near future will be even more nearly perfect than those now in use.

V. Mine-Hoisting.

Thirty years ago, while immense hoisting-plants were in use on the Comstock, they were far from efficient, and were not copied even in miniature on smaller mines. The favorite plant in Colorado in the early days was called the Gilpin county hoist, and consisted of a rope-drum securely fastened at one end to a large wooden pulley connected by a slack belt to a stationary engine running continuously at a slow speed. When the signal was given to hoist, the operator opened the throttle of the single-cylinder engine and brought the necessary work-

ing-tension on the belt by means of a tightener operated by a hand-lever. To-day, these primitive machines, large and small, have all been succeeded by direct- or gear-connected steam-engines, equipped, whenever the tonnage is sufficient to justify the expense, with variable cut-off valve-gear, post-brakes, and every modern improvement.

Steam-hoists, capable of handling from 10 to 20 tons of total load, from depths of from 4,000 to 6,000 ft., at speeds varying from 4,000 to 5,000 ft. per minute, are now not at all uncommon. These immense plants are fitted with every imaginable device for increasing the efficiency, rapidity, and safety of operation, and the skill and artistic ability displayed in the design of some of the later Nordberg creations bring them to a point where they can almost be considered works of art.

In many places where water fit for use in boilers is scarce and electric current cheap, as at Cripple Creek, the electric hoist has almost completely replaced the steam-hoist.

When large electric hoisting-plants were first installed, it was found that the great amount of current necessary to start and accelerate the load brought a very objectionable "peak" on the transmission-line. This difficulty has now been overcome by the Illgner and other similar systems, in which the energy stored in a large rapidly-revolving fly-wheel cuts down, if it does not entirely prevent, the objectionable peak. Safety-devices likewise have been very much improved; and a recent invention, whereby the cage-tender is in constant signal-connection with the engineer, should do much to decrease the number of shaft-accidents.

VI. Underground Tramming.

When mines were shallow, shafts numerous and hoisting-facilities inadequate, hand-tramming was almost universally employed, but with increase in depth came the necessity for better hoisting-machinery and a reduction in the number of shafts, thereby increasing the distance over which ore had to be trammed. Then it was found that a high-priced man constituted a very expensive motive-power for pushing cars; horses and mules were put into commission; and, still later, air- and electric locomotives have come into very general use. There has been much discussion concerning the relative merits of

these two latter systems of underground haulage; but there is no doubt that each has its own field. Where the openings are dry and the roof sufficiently high and firm to carry the trolley-wire insulators, there is no question as to the desirability of using electricity, but where these conditions unfortunately do not obtain, the compressed-air locomotive is an excellent substitute.

VII. Timbering.

Where the ore-bodies do not exceed 10 or 12 ft. in thickness, and have a firm hanging-wall, nothing can exceed the cheapness and simplicity of stulls; but when larger ore-bodies are encountered and timbering is necessary, the system commonly employed is that of "square setting," invented by Deidesheimer and first used on the Ophir mine on the Comstock lode in 1861. Timber is yearly becoming more expensive, does not usually last well underground, and when the ore-bodies are large, especially if there is a tendency to movement in the walls, "square sets" made from square timbers are apt to "swing" and afford very little vertical support.

When large quantities of timber are required for square setting, in situations where the distance from the forest to the mine is not too great, round timbers are very much cheaper and more durable than square timbers. A round log has about double the strength of a timber cut from the inscribed square on its small end; and since, in the round log, the concentric rings of wood-growth are unbroken and each protects the ring immediately underneath it from decay, the comparison, both in cost and in durability, is very unfavorable to square timbers. Automatic framing-machines can now be had which utilize the full strength of round timbers by making a bevel-joint outside of the square tenon necessary in all square-set timbering. This additional segmental contact-area in the joints braces the round timbers so that they are much less liable to "swing" in large stopes than the square.

In some cases, "cut and slice" and "caving" methods are employed, in which the hanging-wall is allowed to come down and rest on each successive floor as the ore is stoped out, and is prevented from mixing with the ore by a mass of crushed timbers and plank which follows down on the top of the receding ore. In the Utah Copper and Boston Consolidated mines,

at Bingham, Utah, about 4,000 tons of copper-ore are mined daily by the "caving" system. In some large mines the stopes are filled with waste as fast as they are freed from ore, and the ground above thereby prevented from caving, in the same way as if stulls or square setting were used. Steel and concrete are coming slowly into use in shafts, stations, and tunnels, and with the natural decrease in the price of iron and cement on the one hand and the rising cost of timber on the other, it is easy to see that the more durable forms of construction will eventually supersede wood on all permanent work.

The direct-replacement system employed in the Rio Tinto copper-mines in Spain succeeded perfectly in holding both walls and surface in place on a vein from 200 to 260 ft. wide, and, by taking advantage of the wonderful skill of the Spanish miners in building dry stone walls, gives a new method of safely and economically mining the lower portions of the lodes which cannot be reached by the open-cast systems extensively in vogue there.

VIII. Pumping.

In the United States, the old-fashioned Cornish pump, with its costly foundation, massive walking-beam, huge plunger-rods and ponderous balance-bob, was supplanted many years ago by the direct-connected steam-pump, which soon developed into a most efficient pumping-machine with duplex, triple-expansion engines and every refinement possible in modern steam-engine practice. These have in many places been superseded by the electric-driven plunger-pump, in which the high speed of the electric motor has been reduced by suitable gearing. Latterly, quite a large number of electric pumps have been built in which the gear is entirely eliminated. The speed of the motor has been reduced, and that of the plunger raised, forming a combination known as the express pump. Pumps of this class, with capacities of 1,600 gal. per minute, raising water 1,550 ft., are now being very successfully employed in unwatering the Comstock Lode at Virginia City.

Within the last few years great improvements have been made in the electric-driven turbine, which, with its entire absence of valves and reciprocating parts, threatens to dominate the field completely.

Already we have single-stage turbine-pumps raising 35,000

gal. of water per minute 150 ft. high; five-stage pumps raising 10,000 gal. per minute 600 ft.; six-stage pumps raising against 800 ft. head; and eight-stage pumps raising 400 gal. 1,400 ft.; and responsible firms are ready to contract to raise water by this system to any elevation up to 2,000 ft., and guarantee a pump-efficiency of from 60 to 75 per cent., according to conditions of service.

IX. Mine-Lighting and Signaling.

Incandescent electric lighting has long since driven oil-lamps and candles out of underground stations and permanent levels in which any large amount of work is carried on. The recently invented tungsten-lamp with its high efficiency, giving 20 c-p. with an expenditure of only 25 watts per hour, makes it economically possible to extend electric lighting very greatly throughout underground workings. In some of the largest and most progressive mines, candles and oil-lamps have already been replaced in the stopes by acetylene-lamps, which are not only cleaner and safer, but give a much greater illumination for a given cost.

For mine-signaling, the flash-light system operated by interrupting, by means of well-protected switches, the current passing through the station-lamps is rapidly replacing the old-fashioned cumbersome bell-cord; and the latest moisture-proof mine-telephones give instant communication throughout the underground workings, and to and from the surface, with little danger of interruption.

X. Explosives.

The use of modern high explosives in mining and tunneling is now universal; but there is a crying demand for an explosive which can be more safely handled, and which on explosion or detonation will produce a smaller amount of noxious gases, which not only injure the health of the miners, but delay the resumption of work after each round of shots has been fired.

Irregularities in the composition of explosives, variations in the strength of detonators, and differences in the speed of fuses are all fruitful sources of mine-accidents; and, while too much "paternalism" is certainly to be avoided, it is doubtful if anything short of governmental regulation and inspection of explosives, detonators, and fuses will ever bring about the uniformity necessary to safety.

XI. Mine-Ventilation.

Less progress has taken place in this department than in almost any other, although the means for moving large quantities of air under slight pressures have been very much improved. The ventilation and cooling of metal-mines have not yet received the attention which their importance demands. In this respect Western engineers could take profitable object-lessons from their brethren in the coal-fields. Very few of our Western mine-operators go to the trouble of recording temperatures and making ventilation-maps, showing the direction of the air-currents, etc., all of which data are necessary before a satisfactory system of either natural or artificial ventilation can be planned. As most of the Western mines are in hilly or mountainous situations, it is generally easy to provide two openings at greatly different elevations, so that the heating-effect of the workings can be depended upon to control the direction of the air-currents to such an extent as at least to cool and ventilate the workings partly. When these advantages cannot be obtained, centrifugal or forced-draft blowers, driven either by steam or by electricity, furnish an easy means of obtaining the desired results.

The latest high-speed electric direct-driven centrifugal compressors give pressures up to 45 oz., and have been built in sizes up to 40,000 cu. ft. of free air per minute; but there is apparently no limit to the size of the machines which can be built under this system.

XII. Tunneling.

As the United States continues to grow in wealth and importance, tunneling-operations increase in like proportion, both in number and in magnitude. New York City and its environs are now underlain by a net-work of tunnels, and other cities are rapidly developing underground systems, since through the increase in population business and travel become congested. All over the United States, water-supply, hydro-electric power and the reduction of grades on railways are requiring new and expensive tunnels, to which, in the West, are added the great irrigation-tunnels called for in both government and private enterprises. The result of all this activity in tunneling has been a vast improvement in both machinery and methods, and

a greatly increased number of thoroughly trained and skilled workmen, so that records formerly unattainable in the United States are now being made in widely-separated localities. Table I. gives the monthly rates of progress reached by some of our latest achievements in tunnel-driving. The recent paper of Mr. Saunders[1] describes systems and results abroad, which show much higher rates of progress than we have yet been able to attain in the United States.

TABLE I.—*Progress in Driving Tunnels in the United States.*

Cowenhoven tunnel, Aspen, Colo., 7 by 8 ft.,	May, 1893, 421 ft.
Roosevelt tunnel, Cripple Creek, Colo., 9 by 10 ft.,	January, 1909, 435 ft.
Gunnison tunnel, Gunnison, Colo., 12 by 12 ft.,	January, 1908, 449 ft.
Elizabeth tunnel, Los Angeles, Cal., 12 by 12 ft.,	October, 1908, 466 ft.

XIII.—GOLD-DREDGING.

Chain-bucket dredging for gold was first attempted in 1867 in Otago, New Zealand, and the first steam-actuated dredge operating on this principle was built on the Molyneux in 1881.

From a few small dredges copied after those in use in New Zealand, gold-dredging in this country has grown into a great industry, which is carried on successfully from the frozen gravels of the Arctic to the sun-scorched river-bars of the tropics, and at all altitudes from 10,000 ft. down to sea-level. Under stress of competition and the necessity of meeting new conditions, dredges have grown both in capacity and efficiency to a point not even dreamt of a few years ago. Dredges are now built with close-connected buckets, of capacities up to 13.5 cu. ft., and capable of handling 10,000 cu. yd. of gravel in 24 hr.; some have been built with bucket-ladders capable of digging 67 ft. below the water-line and 20 ft. above it. A new stacker now under construction will deliver tailings 160 ft. away from and 60 ft. above the deck of the boat. Improved construction and better management have rendered dredging-operations less dependent upon weather; and last winter in Colorado a dredge was operated continuously at an altitude of 9,990 ft. where the temperature on several occasions fell to 20° below zero. The subject of suction-dredges has been fully considered in Mr. Granger's recent paper.[2]

[1] This volume, pp. 432 to 458.

[2] *Ibid.*, pp. 496 to 516.

XIV. Electric Transmission.

No sketch of this kind would be complete without some notice of the immense service which the mining industry is receiving from long-distance electric transmission. While a few mines are favorably situated for the utilization of adjacent water-power, many of the principal mining-districts of the United States are at altitudes so great that any available water-power is far below them. Again, as in the case of Nevada, Arizona, and portions of Utah, the mines occur in an arid country where it is difficult to obtain sufficient water for domestic purposes, to say nothing of power.

Already the electric current is carried to all elevations from sea-level to timber-line, and there is scarcely a desert mining-camp of sufficient size to justify the erection of a pole-line that is not equipped with electric power. The ease with which this overcomes the old and apparently insurmountable problems of scarcity of water and fuel constitutes one of the delights of modern mining.

The use of electricity has also completely solved the old vexed problem of underground sinking and hoisting, so that these operations are now as readily carried on from deep tunnel-levels as from the surface.

Transmission-lines of all lengths up to 220 miles are in daily use, with carrying-capacities ranging up to 40,000 kw. Long-distance transmission-systems are in many cases operated at 100,000 volts, and new lines are building to utilize even higher pressures. Recent improvements in insulation promise to make still higher voltages possible, which would mean a corresponding increase in the distance to which current could be profitably carried.

XV. Sampling.

Few departments of mining engineering have shown greater advance than ore-valuation.

In milling- and concentrating-plants, where fine crushing is a necessary preliminary, sampling is a comparatively easy and reasonably accurate operation, but where the ore is to be treated by blast-furnace smelting, crushing of any kind is objectionable and fine subdivision is prohibited. Forty years ago, for the valuation of coarse ore, "grab" sampling was in common use, and this method was replaced in slow succession by Cornish

quartering, fractional division, and split-shovel sampling. Then came automatic sampling in many forms, but all taking a portion of the ore-stream continuously. In 1884 a new system of sampling was invented which automatically deflects the entire ore-stream for a varying portion (usually one-fifth) of the time into the sample-division. Numerous different machines working on this principle are now in use; and these types of sampling-plants have been perfected to such a degree that where the hopper ore-cars which are now coming into general use are employed, ore may be unloaded, crushed, sampled, and reloaded into the outgoing cars, and the ground sample delivered in a locked steel box, without ever having been handled—the entire chain of operations being performed automatically.

XVI. Concentration.

The separation of valuable minerals from worthless gangue must have been one of the earliest operations in the history of metallurgy. Up to less than 100 years ago the pan, tub, and inclined plane, which are all so graphically illustrated by Agricola, continued to be the only devices in use. Hand-jigs were first introduced for the separation of coarser particles than could otherwise be handled, and the principles involved are in use to-day, although improved mechanical appliances have changed and enlarged operations to such an extent that the primitive origin would scarcely be recognized. About 35 years ago the use of air as a concentrating medium was successfully introduced, and, despite its many disadvantages, this system, assisted by numerous mechanical improvements, still exists and manages to hold its own where water is unobtainable or, for some reason, cannot be used. In skillful hands, some of the pneumatic separators give wonderful results, but the delicacy of the adjustments and the attendant dust will undoubtedly prevent any extensive employment of this method.

The pulsating water-current recently invented by a distinguished investigator in the concentrating field has already won a place for itself in both sizing- and jigging-operations, and promises to become a most important factor in concentrating-work.

Specific gravity is, however, no longer the only principle taken advantage of in mechanical ore-sorting. To these have

been added magnetic and static electric separation, and many different methods based on the surface-tension of water (with or without the assistance of oil or acid), resiliency, and affinity for grease. The latter method, used very sparingly in this country, finds its chief application in South Africa, where it is used in the separation of diamonds from other stones.

The recent discovery by Kunz and Baskerville that the use of ultra-violet light would enable an observer to determine by inspection, with reasonable accuracy, the percentage of willemite in concentration-tailings, has already found commercial application on a very large scale, and opens up a wide field for speculation as to what the future may hold in store for us in this field.

The enormous size of some of the new concentrating-plants erected in the West exemplifies in a marked degree the magnitude of the operations now being carried on. At Anaconda, Mont., the Amalgamated Copper Co. has an eight-unit concentrating-plant, each section of which handles 1,000 tons in 24 hr.; and some of the new plants at Ely, Nev., and Garfield, Utah, are but little smaller in size.

XVII. Roasting for Blast-Furnace Smelting.

The earliest roasting-furnaces to prepare sulphide ores for blast-furnace smelting were small hand-operated reverberatories, with or without fusion-hearths. These were followed by revolving cylinders and various types of mechanically operated reverberatory furnaces, all of which were not only expensive to operate and keep in repair, but yielded a product very badly adapted to blast-furnace work. To-day these old-fashioned furnaces, both hand and mechanical, have been almost entirely superseded by blast-systems like the Huntington-Heberlein, Carmichael-Bradford, and Savelsberg, which make it possible to utilize the many advantages of blast-furnace smelting for the treatment of concentrates, fine ore, and flue-dust at very low preparatory costs. These systems marked a wonderful advance over the old reverberatory roast with its pulverulent product; but they were still open to the serious objection of requiring a large amount of manual labor in charging and discharging the pots and breaking up the sintered product. In spite of this drawback, the results obtained were so desirable that study and in-

vention along these lines have been stimulated to such an extent that there are already in use mechanical roasting- and sintering-plants (system of Dwight and Lloyd), to which ore can be fed in a steady stream, and which will automatically deliver a desulphurized, sintered, and broken-up product in the best possible condition for blast-furnace work.

XVIII. Lead-Smelting.

Progress in this department of metallurgy during the past decade, in the West, has been much hampered by three conditions: (1) all the lead-smelting plants operate almost entirely on custom ores; hence the supply is irregular in volume, grade, and composition; (2) the great extension of the leasing system throughout the West tends to bring ore into the market in very small lots, thereby increasing the difficulties and cost of storage and bedding; (3) eight years ago nearly all of the principal lead-smelting plants in the United States passed into the hands of a corporation organized for that purpose, and the industry was thereby deprived of the stimulus of healthy competition.

The largest lead-furnaces in the United States have hearths 44 by 180 in., and treat daily from 150 to 255 tons of ore, according to its character. Mechanical charging is used in some cases; but while it slightly reduces operating-costs, it is no improvement metallurgically.

In Australia, where different conditions prevail, the improvements in lead-furnaces have kept pace with those in iron- and copper-smelting.

XIX. Reverberatory Copper-Smelting.

In 1867, when Richard Pearce (afterwards President of the Institute) built his first reverberatory furnaces at Black Hawk, Colo., they were considered the acme of metallurgical perfection, and their successful operation did wonders for the mining industry of the State. The hearths of these furnaces were only 8 by 12 ft. in size, and their daily capacity was 12 tons. As the quantity of ore produced increased and the necessity for handling larger tonnages became apparent, the reverberatory furnaces have been steadily enlarged and improved, until to-day they have attained almost incredible dimensions,

having a hearth-area of 19 by 116 ft. and a daily working-capacity of more than 300 tons, which, in the case of easily-smelted ores, has risen to over 400 tons. These large furnaces secure a great saving of heat, and uniform, continuous operation, for reasons into which I need not enter here.

XX. Blast-Furnace Copper-Smelting.

The early water-jacketed blast-furnaces for smelting copper-ore were small, round, wrought-iron affairs, about 30 in. in diameter, and rarely smelted more than 12 tons of ore in 24 hr. From this puny beginning, keeping pace with the rapidly-growing copper industry of the United States, furnaces have grown steadily in size and improved in mechanical construction, until they have reached the enormous dimensions of 87 ft. in length by 4 ft. 8 in. in width, with a daily smelting-capacity of 3,000 tons of charge. These furnaces are mechanically fed, work under an air-pressure of 40 oz., and give infinitely less trouble than their smaller progenitors.

The water-jackets are completely sectionized, and it is possible to renew most of the sections without stopping the furnace. At first only one tier of jackets was used, then two tiers came into use, and now some of the most recent furnaces have air- or water-jackets replacing the brick superstructure, thus doing away with much of the roof-accretion nuisance. On furnaces built with a crucible, baby water-jackets have replaced the old cast-iron plates, and water-jacketed nose-pieces have greatly lengthened the life of the furnace-discharge spouts.

The credit for the latest improvements in design and increase in size of both reverberatory and blast-furnaces is due principally to E. P. Mathewson, whose untiring energy as an investigator and skill as a metallurgist have made the Washoe plant, at Anaconda, the Mecca for progressive engineers from every country.

The recently-invented acetylene blow-pipe promises to be of great service to blast-furnace engineers, as through its use it will soon be possible to obtain welded water-jackets entirely free from the objectionable lap-seams and rivets.

XXI. Electrical Smelting.

A great amount of experimental work has lately been carried on by engineers in various parts of the world having for its

object the utilization of the electric current in metallurgical operations, and there is no doubt that in the near future many minerals now smelted with fuel will be reduced to metals by electrical processes. The electric current possesses the great advantage of allowing a most efficient utilization of the heat: and also complete control of the exposure of the molten metals to air or gases.

XXII. Briquetting.

At many blast-furnace plants "fines" are made into briquettes with the ordinary die-and-plunger machines; but if free acid or copper sulphate be present, the surfaces of both dies and plungers are rapidly corroded, which obviously increases the diameter of the dies and diminishes that of the plungers. As this solvent action continues, a point is soon reached when the plunger no longer fills the die-opening, and pressure forces the material through the space between them, instead of consolidating the mass.

When a sufficient amount of plastic material, such as slimes, can be obtained to mix with fine ore and flue-dust, so as to give the mass the property of "flowage" under pressure, it may be briquetted in machines similar to those used in making building-brick by the "stiff-tempered" process. Solvents do not interfere with the operation of these machines, and by constructing the working-parts of steel and phosphor-bronze, it is possible to make briquettes dry enough to pass directly into a blast-furnace at the rate of 600 to 800 tons per day for each machine employed, at a cost of less than half that of the die-and-plunger system.

XXIII. Chlorination.

Chlorination is confined almost entirely to sulphide gold-ores carrying so little silver that its loss may be disregarded, and which require a preliminary roasting before treatment. The early methods of tank-leaching have all been superseded by barrel-chlorination, and in some of the latest plants chlorine is produced by electrolysis instead of by the decomposition of bleaching-powder with sulphuric acid.

The largest plants operating under this system treat custom ores, and the stress of competition, together with the necessity of handling constantly increasing tonnages, has brought about great improvements in both machinery and methods.

The most desirable features of this process of gold-extraction are the high percentage recoverable, and the rapidity with which clean-ups can be made, rendering it easy at all times to know exactly what results are being obtained.

The largest plants operating under this system are situated in Colorado City, Colo., where two mills owned by one concern have an aggregate capacity of 800 tons per day.

XXIV. Cyanidation.

While the first patent for extracting gold from its ores by cyanide solutions was issued in 1867, it was not until McArthur and Forest took it up in 1889 that practical results of any value were obtained. Since that time the use of the process has increased by leaps and bounds in all of the principal gold-producing countries, and to-day it is the principal factor in the world's steadily increasing gold-production. Cyaniding seems to work with equal facility on raw ore, roasted ore, and tailings, and with the steady improvement in the mechanics as well as the chemistry of the process, it bids fair to do even greater things in the future than it has done in the past. Nor is its use confined entirely to the extraction of gold. In many districts it is operating with great success on mixed gold- and silver-ores. At Millers, Nev., two plants with an aggregate capacity of 700 tons per day are operating on Tonopah ores, in which the average ratio of silver to gold is 80 to 1.

The largest and perhaps the most complex cyanide-plant in the United States is the Golden Cycle mill, at Colorado City, Colo., which has a daily capacity of 1,000 tons, and treats exclusively Cripple Creek ore, all of which requires careful roasting and very fine crushing. The Homestake cyanide-mill handles a larger tonnage, but treats only tailings.

Next to this plant in point of size and importance, and handling ore of much higher grade, is the new mill of the Goldfield Consolidated Co., Nevada, which has a capacity of 600 tons per day, and contains the very latest improvements, culled from American, African, Australian, and Mexican practice.

XXV. Fume-Recovery.

No department of metallurgy has made slower progress than this most important division, but now, through an unholy alli-

ance between the unscrupulous contingent-fee attorney and the greedy land-owner, the success of "smoke-farming" is compelling the smelting companies to do for self-preservation something which they should long ago have undertaken for profit. Out of the almost numberless devices which have been tried for fume-recovery, the bag-house affords the best solution of the problem yet devised. Bags were used for the recovery of both zinc oxide and lampblack more than 50 years ago, but were not used for fume-recovery until 1878, when Bartlett set up a small plant at Portland, Me. The first large successful installation of this kind was erected by the Globe Smelting Co., in 1885, and has been in continuous operation ever since. While the bag-house has been eminently successful in the recovery of blast-furnace fume, it cannot be used on fumes from reverberatory roasting- or smelting-furnaces, owing to the fact that a portion of the sulphur dioxide formed by the oxidation of the sulphur is raised to sulphur trioxide by contact with incandescent ferric oxide. At the United States Smelting Works, in Utah, provision for protecting the bag-house from sulphur trioxide is made by blowing into the flues zinc oxide, which immediately absorbs the sulphur trioxide present to such an extent that it has been found quite possible to use cotton bags. The bag-house has proved very efficient in recovering fume from lead-refineries and all lead-smelting operations excepting roasting. In large copper-smelting plants, where large volumes of gas are produced carrying a sufficient amount of sulphur trioxide to destroy woven fabrics, three systems are in use—namely: radiation, decreased velocity, and friction—and in many cases two or more of these methods are combined. At the Washoe plant, in Anaconda, long steel-covered flues of enormous cross-section have been installed for a number of years, and, while this system does not effect a complete recovery of the fumes, it is as nearly perfect as it is possible to make a plant to-day. At Great Falls, Mont., the Boston & Montana Co. is installing the friction system at a cost exceeding $1,000,000, which includes the construction of a stack 506 ft. high and 50 ft. in diameter and a dust-chamber in the flue-system, of such width that the furnace-gases will pass through it at a velocity considerably less than 500 ft. per minute. From the roof of this flue-chamber more than a million steel wires will

be suspended, an arrangement which experiment has shown to increase greatly the settling-efficiency of the dust-chamber. This installation is practically completed, but it will be some time before the results obtained can be accurately determined.

I need not add that where (as in some Eastern works) metallurgical and commercial conditions permit the manufacture of suphuric acid from the fumes, this method offers special advantages.

XXVI. Conclusion.

The subjects already alluded to occupy but a small portion of the field covered by our 4,000 widely-scattered members, but I hope that enough have been mentioned to serve as topics for discussion at this meeting. The ever-widening range of operations, the constantly expanding magnitude of mining undertakings, and the continually increasing complexity of both machinery and methods are daily creating new openings for mining engineers. To meet this demand our technical schools and colleges are yearly sending out an increasing number of graduates, whose opportunities and responsibilities will be even greater than those of the engineers controlling the activities of to-day. Even now, one change very much to be desired is beginning to become apparent. Heretofore it has too often been considered that an engineer's accountability ended when he discharged his full duty to his employer. To-day we are beginning to realize that the public forms a third party, vitally concerned in the results of the work in which mining engineers are engaged. As large investments are usually held by divided ownership and stocks are often scattered far and wide, so that the owners of small holdings have little or no opportunity to become conversant with the exact conditions of the properties the stocks represent, an engineer's duty should be to see that no word or act of his can be construed so as to give one man an opportunity to take advantage of, or mislead, another. Every one, no matter what his station, has a duty to society and his fellow-men which can never be either ignored or neglected. The employer, whether an individual or a corporation, is entitled to all of the information and data which experience, diligent investigation, and careful study can bring to light; and while an engineer has a right to state probabilities from both

indications and analogy, he should never assume the gift of prophecy and thereby delude both himself and others.

Specific information gained in examination or research for one client should never be utilized for the benefit of another, unless there is no possibility that such use will in any wise injuriously affect the interests of any previous employer. This restriction applies with full force to the dealings of the engineer himself, as he should always remember that information gained by him at the cost of another, no matter how laboriously it has been obtained, belongs to the party who paid for it. No matter what success ability, industry, or chance may bring to an engineer, his career has been an absolute failure unless he can truthfully say in his heart of hearts: "No man is poorer because I am richer."

The Ruble Hydraulic Elevator.

BY J. McD. PORTER, SPOKANE, WASH.

(Spokane Meeting, September, 1909.)

In many of the old placer-mining districts are still to be found large tracts of gold-bearing gravel not suitable to be worked with a dredge, because the bed is too shallow or the gulch too narrow. Frequently there is not enough grade to handle the gravel successfully by ground-sluicing or a bed-rock flume, or it contains too many boulders to be worked successfully with the ordinary hydraulic pipe or tube elevator.

In southwestern Oregon, two practical placer-miners named Ruble, after working for years trying to make money out of placer-ground containing many large boulders, invented and patented a hydraulic elevator of an entirely new type, and one that has been found to work very successfully in flat ground and in gravel containing many large boulders. It is a very simple contrivance.

A few years ago I acquired the property near Pierce City, Idaho, known as the American placer-mine. Various attempts had been made to work this ground. A bed-rock flume had been installed by one company, an Evans elevator by another, and still other methods were tried on a smaller scale. All were

unsuccessful. I installed a Ruble elevator, and it has proved very satisfactory. Working under 100-ft. (pipe) head, the ground has been handled at a cost of a little less than 8 cents per cubic yard. The conditions at the American mine are exceptionally hard, the boulders being large, heavy, and numerous. Basalt boulders, up to a size of 16 by 18 by 32 in., have been elevated to a height of 16 or 17 ft., with a 4-in. nozzle-stream, under 100-ft. head. Boulders larger than this size are blasted.

At the American mine, a foreman, two pipemen, and two laborers are required per day of 24 hr. to operate the elevator. Several sizes of this elevator are in use, the one at the American mine being the 25-ft. size, having 25 ft. of grizzly 8 ft. wide. This elevator handles from 280 to 300 cu. yd. per day. In ground containing fewer and smaller boulders the capacity would be much greater and the cost of operation much less per day and per yard. At first I used two pipemen on each shift, one driving the gravel to the elevator, the other elevating it, each using a No. 2 giant with 4-in. nozzle. Later, I changed to No. 3 giants, using 5.5-in. nozzles, which enabled me to use all the water in one stream, dispensed with one pipeman on each shift, and materially reduced the cost of operation. I also found that I could handle about 25 per cent. more yardage in this manner than by dividing the water into two streams.

The elevator is so constructed that it is impossible to choke it. The elevator giant, or the giant that elevates the gravel, is placed in line with, and about 50 ft. from, the elevator, which leaves room for the field giant to pile up gravel in front of the elevator. The water is then shut off from the field giant, turned into the elevator giant, and the gravel run through the elevator. While this is being done, the field giant may be moved and reset, if necessary. In deep, undrainable ground, a water-lift is attached to the side of the elevator, to carry off all the water after using. The foreman and laborers can reset the giant while the pipeman keeps the water continually at work. The sluice-boxes may be cleaned up in about 2 hr., while the water is being used in the field giant. The laborers break up the large boulders, cut and burn brush, move and set the giants, help move the elevator, and make themselves generally useful.

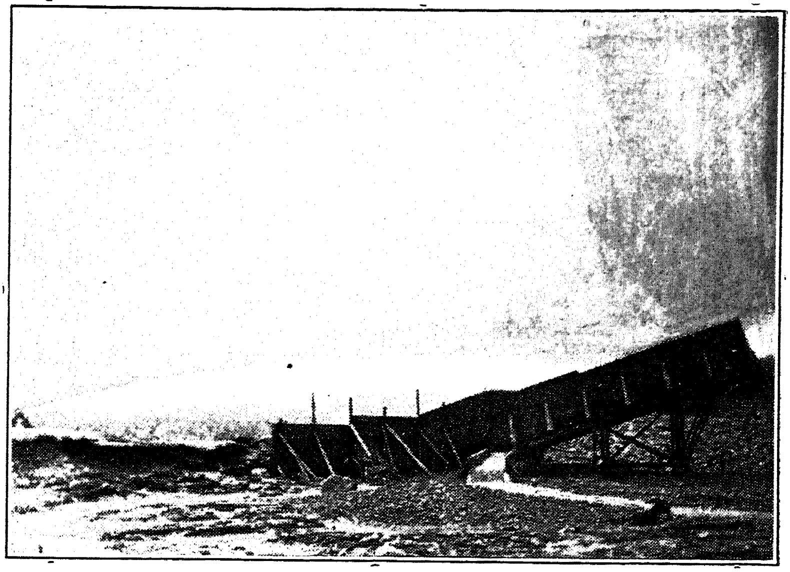


FIG. 1.—THE RUBLE HYDRAULIC ELEVATOR AS OPERATED WITH ONE GIANT.

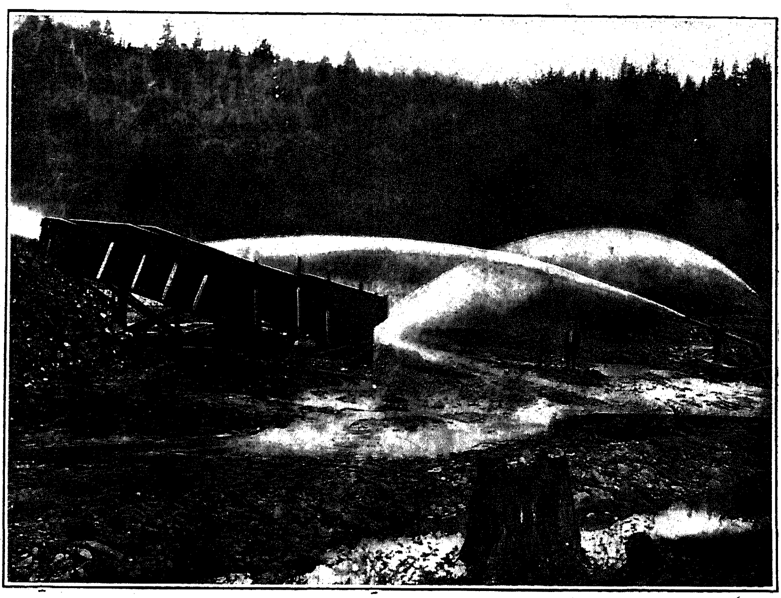


FIG. 2.—THE RUBLE HYDRAULIC ELEVATOR WITH TWO GIANTS IN OPERATION.

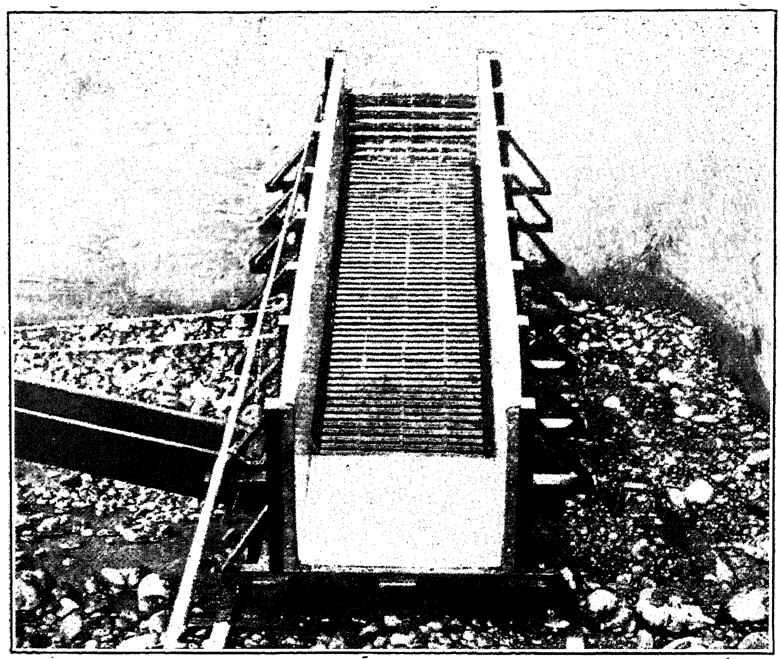


FIG. 3.—GRIZZLY USED WITH THE RUBLE HYDRAULIC ELEVATOR.

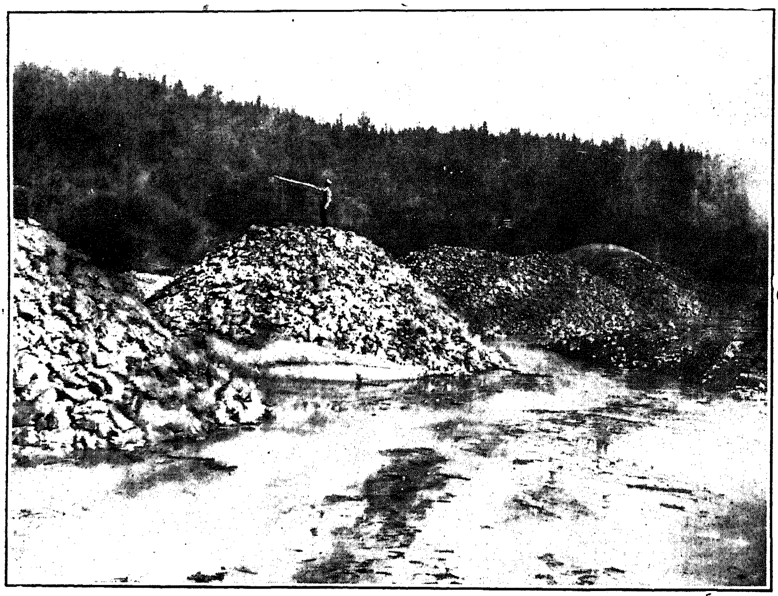


FIG. 4.—BOULDER-DUMPS LEFT BY RUBLE HYDRAULIC ELEVATOR.

The Ruble elevator is a combination of grizzly, undercurrent, and elevator. It separates the fine gravel from the coarse rock and boulders while in transit up the elevator, and does it perfectly. The fine gravel, sand, and gold drop through the grizzly into an apartment underneath the grating-floor, out of reach of the force of the hydraulic stream which is lifting the heavy rocks to the dump. The fine gravel, with its contents, is now acted upon by the water of the elevator giant, after it has lost its force, and with this water is delivered by means of an inclined, smooth floor under the grizzly to the sluice, which is the gold-saving department of the elevator. The fine gravel and sand, after passing the sluice, is delivered to a dump separate from the boulder-dump. Operating with fine material only, the sluices are closely and finely riffled, and are much wider than the usual sluice used for coarse rock, thus furnishing a large gold-saving area in a short sluice.

This appliance solves in an inexpensive and economical manner the difficult problems of inadequate dump, deep and undrainable ground, and the handling of heavy boulders and wash. It effects a close saving of gold and an economical use of water and time.

The elevator rests on rollers, placed on skid-poles on the bed-rock, and is easily moved. When the gravel is washed from in front of the elevator and the space in the rear is filled with boulders and tailings, the bed-rock is cleaned up, a horse is hitched to the elevator, and it is moved forward to a new position nearer the gravel-bank, leaving room for a new boulder-dump in the rear. When horses are not available, the elevator can be moved by hand with a capstan. At the American mine the elevator is moved three or four times each month. It requires from 1 to 1.5 days to move and reset it. The sluice-boxes are usually cleaned up every second day. No time is lost in cleaning up, as the field giant is kept at work during that time.

A few of the points of merit of this elevator are:

1. It handles larger rocks, with less water, than other types of elevators.

2. It dispenses with the boulder-crew, either at the mine, ground-sluice, sluice-box, or dump, except in rare cases.

3. It saves the gold, and the gold-saving department may be protected by lock and key.

4. The gravel is picked up in close proximity to the giants, the gold immediately extracted, and the boulders and other waste material dropped back on bed-rock previously worked off, instead of being transported a long distance to the dump. It makes its own dump.

Strange as it may seem at first thought, it is easier to elevate the gravel than it is to drive it along the level bed-rock, as is shown when the water is divided into two equal streams, the field giant being unable to keep the elevator giant supplied with gravel. In driving the gravel along on the bed-rock, the water has more or less downward pressure, causing more friction, while on the approach and the grizzly, owing to the inclined construction, it is lifting the gravel and boulders, causing less friction or dragging.

Fig. 1 shows the operation of the elevator with one giant, the sluice discharging in the foreground.

Fig. 2 shows the elevator with two giants in operation.

Fig. 3 shows the interior appearance of the grizzly; also the sluice swung from its bearings, with the apron or approach removed in order to move ahead.

Fig. 4 shows boulder-dumps left by this elevator at the American mine.

*

Modern Practice of Ore-Sampling.

BY DAVID W. BRUNTON, DENVER, COLO.

(Spokane Meeting, September, 1909.)

FROM the old-fashioned "grab-sample" to the modern timing-device, which takes a machine-sample with mathematical precision, there is a wide gap, which was only crossed by many years of toil and unremitting endeavor. Even to-day, notwithstanding the advancement in the art, "grab-sampling" is still practiced—sometimes to afford the unscrupulous mine-promoter a basis for fairy-tales with which to entrap the too-gullible investor, and often by milling and smelting companies to determine the amount of moisture in custom-ores. The latter practice is almost as reprehensible as the former, and it causes more trouble and ill-feeling between seller and buyer than all other factors put together. No reputable concern to-day would think of attempting to determine by grab-sampling the amount of gold, silver, lead, or copper contained in an ore, and yet many buyers expect the miner to accept the results of grab-sampling in the determination of the amount of water contained in the ore, forgetting that accurate results are just as necessary here as in the determination of the metals, because the result determines the percentage of weight of the ore which shall be excluded and considered to have no value whatever.

Samples for the determination of moisture should be taken with as great care as samples for the determination of metallic content, and in order to avoid the extra expense of a separate operation moisture-samples should be taken from the sample-safe. As the sample reaches the sample-bin in a smaller stream and by a more circuitous route than the "reject" travels in its path to the outgoing car, it loses more moisture *en route*, and a constant should be added to compensate for this difference. Carefully-conducted experiments have shown that the difference in loss of moisture between the two routes does not exceed 10 per cent. in summer and 7 per cent. in winter. For instance,

a lot of ore shipped during the summer months, in which the machine-sample showed 5 per cent. of moisture, would have an actual moisture-content of 5.5 per cent. Grab-sampling by an interested party, at its best, is only a prejudiced conjecture, while at its worst it gives rise to the most unscrupulous practices with which the ore-producer and the mining-investor have to deal.

Shovel-sampling, another archaic method which is still used in some localities, consists in throwing out from the car or wagon every third, fifth, or tenth shovelful for a sample. As the portion of the pile from which the sample is taken is entirely at the discretion of the operator, the process would be more properly named fifth-shovel selection than fifth-shovel sampling. Between the conscientious workman who endeavors to be absolutely upright, and often becomes, as the Scotchman said, "maer than plumb," and the scheming laborer who, desiring to make his "job solid," takes a "safe sample," there is little room for truth or accuracy in this method, and the sooner it is consigned to oblivion the better for every one concerned.

Thirty years ago Cornish quartering was the almost universal method of sampling in use, and it is still employed to a considerable extent in cutting-down machine-samples and in mine-examinations where no machinery can be had. When properly carried on with skill, care, and common honesty, fairly-good results may be obtained by quartering, but between the possibility of accidental mistakes and the opportunities which it affords for skillful and unscrupulous operators to manipulate the sample, it has fallen almost into disuse, and should have been completely abandoned long ago. The inherent defect of this system lies in the fact that piling a lot of ore in the form of a cone does not mix it, as the advocates of this system claim. Dropping shovelful after shovelful of ore on top of a cone, instead of building up a homogeneous pile, actually produces a very perceptible sorting-action, whereby the fines build up where they fall on the center of the cone and the coarser particles roll outward and down the sides. This is illustrated in Fig. 1, which is a half-tone from a photograph of a cone built up in actual sampling-practice and bisected by a sheet of glass. This section shows conclusively the great difference in the relative proportion of coarse and fines between the outer and

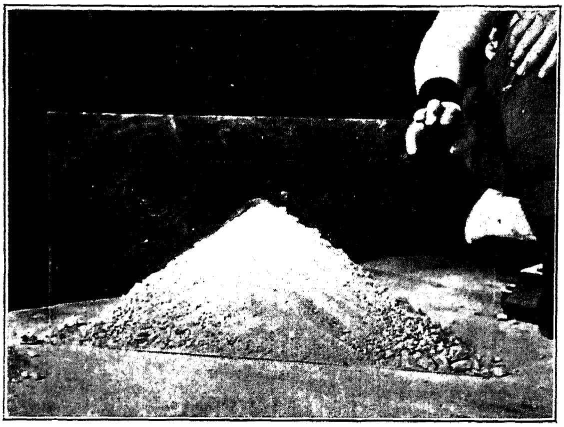
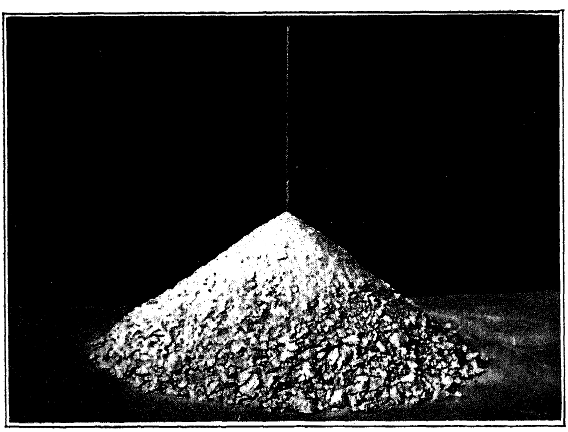


FIG. 1.—SAMPLE BISECTED BY A SHEET OF GLASS, SHOWING PROPORTION OF COARSE AND FINES.

FIG. 2.—SAMPLE SPREAD OUT INTO A "PANCAKE."



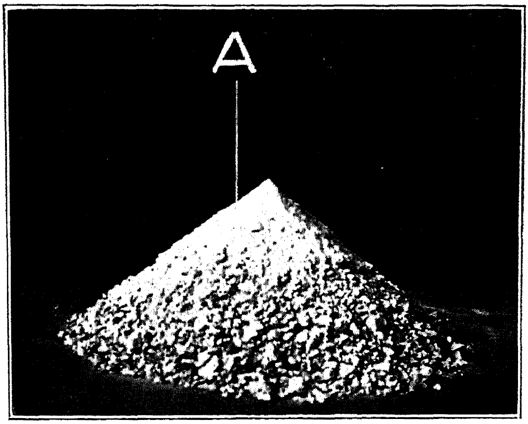


FIG. 4.—SAMPLE CONE WITH DRAWN CENTER.

FIG. 5.—SAMPLE WITH DRAWN CENTER SPREAD OUT.

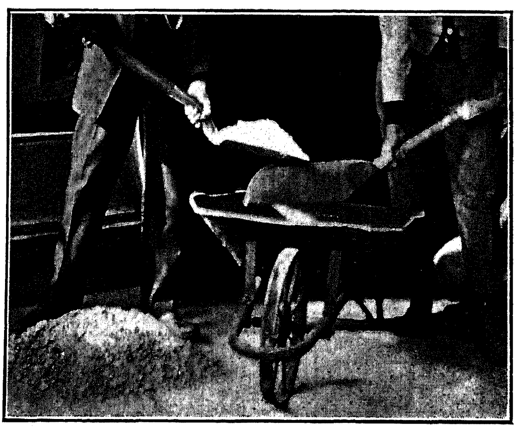


FIG. 6.—U-SHOVEL SAMPLING.

inner portions of the cone, and also makes it perfectly clear that even after the cone has been spread out into a "pancake," as shown in Fig. 2, the fines in the lower portion of the cone will be entirely undisturbed. The most uniform and best results are obtained by coning around a rod, as shown in Fig. 3. By this means the center of the cone is maintained in a vertical line, and if care is taken in working down the cone to a "pancake," as shown in Fig. 2, and separating the quarters by steel blades, so that there is no difference between the quadrants taken for the sample and those thrown into the reject, the results give a fair approximation of the truth, though it is not possible to duplicate results very closely by this method, even at its best.

It would take altogether too much space here to enumerate the different schemes which unprincipled operators have introduced into this method for the purpose of "throwing" the sample, and description of one of them will suffice.

The most ingenious of these plans, and one which is so difficult to detect that it can be carried on directly under the eyes of a skilled observer without detection, is what is known as "drawing the center." The cone is started on the floor, as shown in Fig. 3, but without any rod to determine the position of the center. The operator in charge of the work, in dropping his shovelfuls of ore on the top of the cone, does it in such a manner as to draw the center of the cone imperceptibly in a certain direction, so that by the time the entire sample is piled and ready for spreading, the apex of the cone, shown in Fig. 4, is several inches, we will say, to the SE. of the original center, which is indicated by the perpendicular line, *A*. The ore may now be spread as usual with shovels or with a board, and cut and marked into quadrants by steel blades in alignment with the four points of the compass, as shown in Fig. 5, where the rod, *A*, indicates the original center of the cone, which, of course, has been entirely undisturbed by the mixing and spreading of the upper portion. By rejecting the NW. and SE. quarters an excess ratio of the fines is eliminated, and since these are generally the richest ore, the metallic contents of the two retained quadrants, shown in Fig. 5, will be somewhat less than the average of the original pile. Suppose a 2,000-lb. lot is to be reduced to 62.5 lb., it would mean that the "quartering"

(really halving) would have to be repeated five times: and if at each stage the sample taken represented 98 per cent. of the actual value of the cone, the final sample would only give 90.3 per cent. of the true value of the cone, as shown in the following tabulation:

	Original Lot.	First Cut.	Second Cut.	Third Cut.	Fourth Cut.	Fifth Cut.
Weight, pounds, . . .	2,000	1,000	500	250	125	62.5
Percentage of true value, .	100	98	96	94.1	92.2	90.3

The shifting of the cone-center is easily carried out; in fact, it is difficult to avoid it unless some definite means of preventing it is adopted. Fig. 1 shows very clearly the structure of a cone with a "drawn" center, and in this instance the effect was entirely unintentional.

The irregularities in the results obtained by Cornish sampling, together with the cost of operation and the amount of room required, soon brought about what is known as "split-shovel" sampling, in which the ore is thrown from a broad shovel, handled by one operator, upon a narrow U-shaped shovel, held by another workman, usually directly over a car or wheelbarrow, as shown in Fig. 6. This method, while it requires two men to do what normally appeared to be the work of one, was cheaper than Cornish quartering, but it proved no great improvement over the latter in point of accuracy, since carelessness in almost any direction interferes seriously with the results.

The earliest attempts at mechanical sampling were made by subdividing a falling stream of ore; a process based on the supposition that an ore-stream could be mixed so as to be perfectly homogeneous. Both analysis and experience have shown that this ideal condition is impossible, and mechanical devices for taking a portion of the ore-stream all of the time have been almost entirely displaced by machines designed to take all of the ore-stream for a portion of the time. It is not practicable to produce a stream of ore which shall be continuous in value through every part of its length, any more than it is possible to produce a stream of ore that is constant in value throughout its width; but by taking a small sample entirely across a falling stream at very short intervals it is found that, while no single cut would give an exact representation of the composition of the entire lot, the average of thousands of these small samples

is so nearly correct that results can be duplicated within very narrow margins, or, in other words, that individual errors are balanced. This was not the case with the devices used for taking a portion of the stream all the time, since the errors due to feed, inclination of spouts, or wear on the bottoms of the spouts are constant, and do not vary during the time the samples are being taken.

Almost coincidental with the discovery of the fact that accurate samples could be obtained by taking all of the stream for a portion of the time, came a very considerable improvement in rock-crushing machinery, so that the modern engineer has a much better opportunity to construct a satisfactory plant than the builder had 20 or 30 years ago. Not only are the rock-breakers and rolls of to-day greatly improved in design, but the manufacturers have availed themselves of modern cheap steel to give all parts an excess of strength over any possible strain, while the use of alloy-steels for the wearing-surfaces permits the machines to be kept in much better repair, and requires fewer stoppages for renewals. For sampling-work, crushers and rolls can now be had which are almost as well made as the ordinary steam-engine, and so designed as to give complete accessibility for renewals and for cleaning.

Gyratory breakers of the Gates type have the advantage of delivering a very uniform product, and in crushing ores that are hard and dry this type forms by all odds the best initial crushing-machine; but with ores that carry wet clay, slate, or other substances which will "pack," it is necessary to use a swinging-jaw crusher, preferably of the Blake type. Rock-breakers may be set to crush to any desired fineness, but it has been found that too great a reduction in the size of the product very materially reduces the capacity. In large crushers it is not usually advantageous to attempt to crush below 2 in. in size.

First-class rolls are now always belt-driven, which eliminates the noise and danger attending the operation of the old-fashioned trains of gears. The best practice in roll-crushing is to crush not smaller than half the diameter of the particles fed to any given machine. This rule gives approximately the maximum crushing-capacity with the minimum production of fines and the lowest expenditure for power and metal. Rolls

require a steady feed, and one which is uniform across the entire width of the shell; consequently, nearly all modern rolls are equipped with some feeding-device. In sampling-mills the shaking-tray is generally used on account of the ease with which the rate of feed can be inspected, and the great facility with which such feeders can be cleaned after each lot of ore has been run.

For fine-grinding machines, the coffee-mill type still successfully holds its own against most of the newer devices, although the modern sample-grinder is much heavier, better built, and more easily cleaned than its predecessors.

The first mechanical samplers were imitations of Cornish quartering, the "whistle-pipe" being the most common type. With ore finely crushed, fairly dry, well mixed, and entirely free from strings and rags, and with the dividers new and exactly centering the pipe, fairly good results could be obtained by this method; but as these conditions never existed in practice, and as the edges of the cutters wore rapidly, thereby moving the dividing-line back from the center, this form of sampling-machine was soon discarded, and I believe has now fallen into absolute disuse.

Following the whistle-pipe sampler came the various forms of mechanical split-shovels; but as there was no place in a spout, no matter how wide or carefully built, where a single U-shaped spout could be placed to take a sample which would represent the entire width of the stream, this form also was soon discarded.

More recently this splitter has been revived by an adaptation of the ordinary hand-operated splitter (see Fig. 12), in which numerous small spouts are so arranged across the entire width of a larger one that the main ore-stream is divided into a great number of smaller ones, the even numbers being deflected to the right and the odd numbers to the left. This plan works very well on the first division, but as it effects a reduction of only 50 per cent. in the volume, the operation must be carried further, and the streamlets forming the sample centered into a broad stream, which, in turn, passes over another set of splitters, the operation being repeated as often as necessary to reduce the sample to the desired size. It has been found, however, that the mixture of the streamlets after their union is far

from perfect, and that there is a considerable difference in the amounts of coarse and fines taken by the sample side of the second cutter, depending on its position relative to the cutter above. If the sample-compartments in the second cutter are directly below the sample-compartments in the overlying splitter, they receive the centers of the streamlets, while the "spread" passes into the reject, and the sample at each step in the bank of cutters receives an amount of fines slightly in excess of the average, thereby seriously affecting the value of the sample, provided there is, as is usually the case, a difference between the metallic contents of the coarse and the fines. Conversely, when each cutter in the bank is placed so as to take the "spread" from the cutter above it, the sample will have less than its due proportion of fines. This disadvantage could be obviated by placing a shaking-tray between each set of dividers, or perhaps even better, by moving one divider horizontally across the other, so that each set of cutters would take all parts of the streams from the cutters above them. This arrangement, however, would require considerable head-room, and give a machine which would make a large amount of dust—a feature which is always objectionable in a sample-room.

The latest types of samplers are designed to overcome the difficulties just described, and are usually known as "time-sampling machines," from the fact that they deflect the entire stream into the sample-compartment for a varying portion of the time, depending on the percentage of sample required. Treating the falling stream of ore as a ribbon, they cut sample-sections directly across its entire width, these portions varying in shape and size with the mechanism employed. Of the many types that have been invented and patented, only three have come into general use, and Fig. 7 shows the shapes of the sample-sections taken by these three machines.

A represents a sample cut from the falling stream of ore by the Charles Snyder 20-per cent. sampler, with four radial intake-spouts, making 7.5 rev. (or 30 samples) per minute; delivery-spout 5 by 25 in. (This sampler is not to be confounded with the Snyder sampler.) It will be seen, on this machine, that an attempt has been made to combine the old-fashioned continuous sample with the time-sampling system by arranging

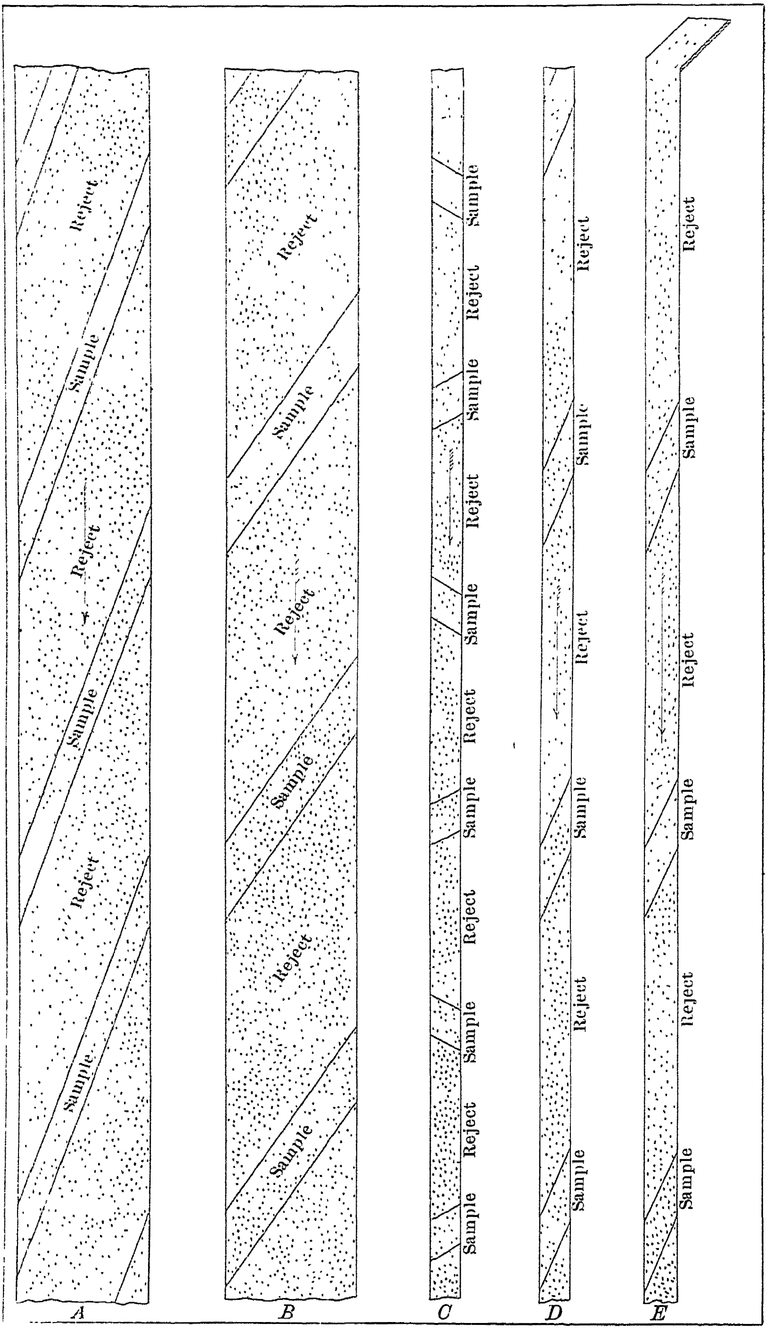


FIG. 7.—SHAPES OF SAMPLE-SECTIONS TAKEN BY THE CHARLES SNYDER, BRUNTON, AND VEZIN SAMPLERS.

the delivery-pipe and intake-spouts so that as one intake-spout passes out of the stream another enters it on the opposite side.

B represents a Charles Snyder 20-per cent. sampler, with two radial intake-spouts, taking 15 samples per minute; delivery-spout 5 by 25 in. This machine does not take a continuous sample, but has the advantage that the intake-spouts, for a given percentage of sample, have double the width, and are therefore much less liable to throttle or choke; at the same time there is no reason why the sample should not be as accurate as that taken with the other type of Charles Snyder sampler.

C represents the sample taken by the Brunton 20-per cent. sampler, taking 54 samples per minute; delivery-spout 5.75 by 5.75 in., cutting-edges parallel.

D represents the sample from a Vezin 20-per cent. sampler with two radial intake-spouts, taking 30 samples per minute; delivery-spout 6 by 6 inches.

E shows the sample taken by a modified form of sector-sampler, which, often through accident and sometimes by design, has come into too-general use.

Both the Charles Snyder and the Vezin samplers have sector intake-spouts revolving on a vertical axis, the only difference between the machines being that the delivery-spout in the Snyder sampler is an annular quadrant, *D*, in Fig. 8, while the Vezin delivery-pipe is either square or rectangular, *E*, in Fig. 9. In order to take a correct sample the cutting-edges of the sector intake-spouts on both of these machines must be exactly radial, as shown in Figs. 8 and 9, otherwise they will include more degrees of arc at one part than at another; and consequently the percentage of sample taken from all parts of the delivery-pipe will not be the same, as is shown by Fig. 10, in which the cutting-edges are not radial to the center of rotation. This, while by no means an exaggerated example of this form of distortion, shows a 74/360, or 20.8-per cent., sample taken on one side of the ore-stream and 88/360, or 24.4 per cent., on the other. If the falling stream of ore were perfectly homogeneous this arrangement would not make any difference, but it is well known that the ore-stream is not uniform, especially in an inclined spout, in which the coarse, rapidly-moving particles go bounding along on the top, while the finer portions

hug the bottom, and on leaving the spout the coarse is projected a considerable distance and the fines drop almost vertically, which gives a sorted falling stream, with coarse on one side and fines on the other. With a tangential feed to a sector intake this sorting-action does not seriously affect the sample if the delivery-spout is perfectly level and free from ridges which would deflect the particles across the stream; but when a radial feed is used, as shown in Fig. 9, and the intake sample-

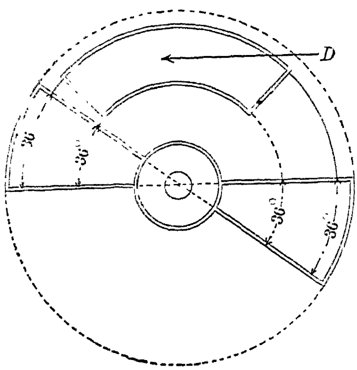


FIG. 8.—DELIVERY-SPOUT OF CHARLES SNYDER SAMPLER. CUTTING-EDGES RADIAL.

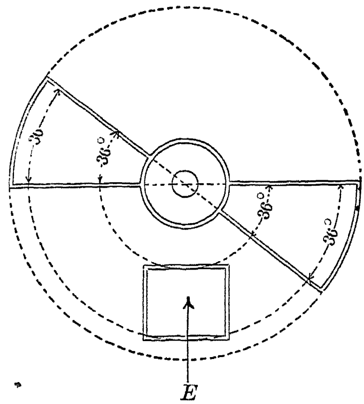


FIG. 9.—DELIVERY-SPOUT OF VEZIN SAMPLER. CUTTING-EDGES RADIAL.

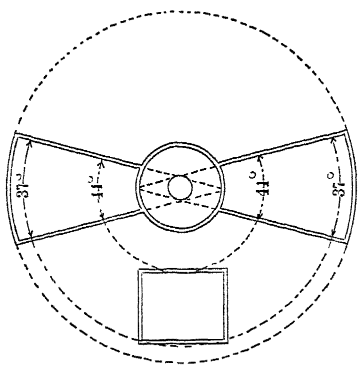


FIG. 10.—CUTTING-EDGES NOT RADIAL.

spout edges are not radial, as shown in Fig. 10, it will readily be seen that a larger proportion of coarse than of fines is taken into the sample.

Since the cutting-edges of this class of samplers, Fig. 11, are necessarily maintained in a horizontal position, they are very liable to become overhung with strings, burnt fuse, and

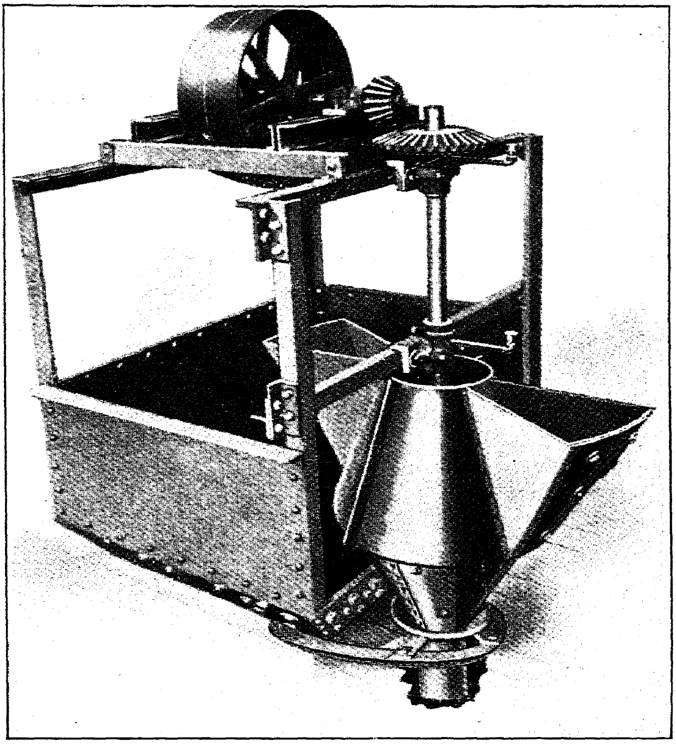


FIG. 11.—VEZIN SAMPLER, SHOWING HORIZONTAL CUTTING-EDGES.

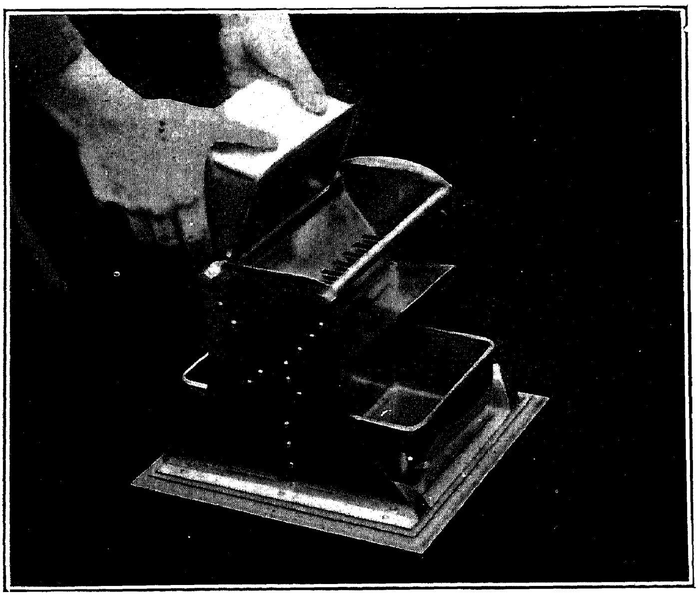


FIG. 12.—TAYLOR & BRUNTON SPLITTER.

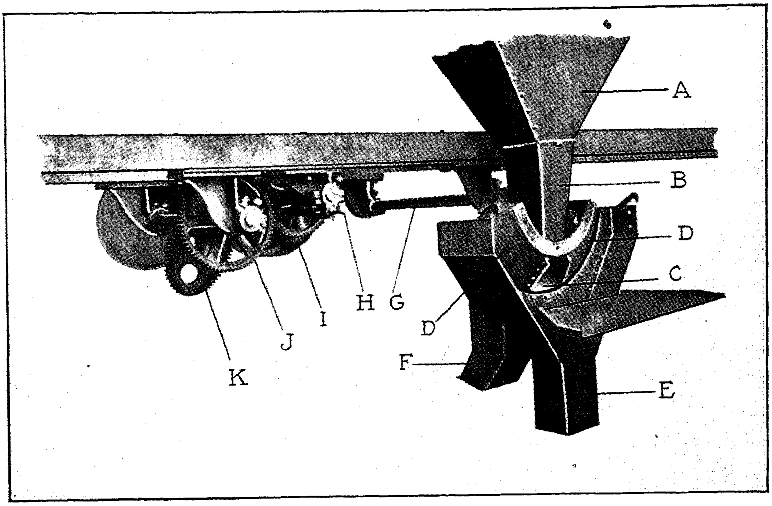


Fig. 13.—The Brunton Time-Sampler. Front View.

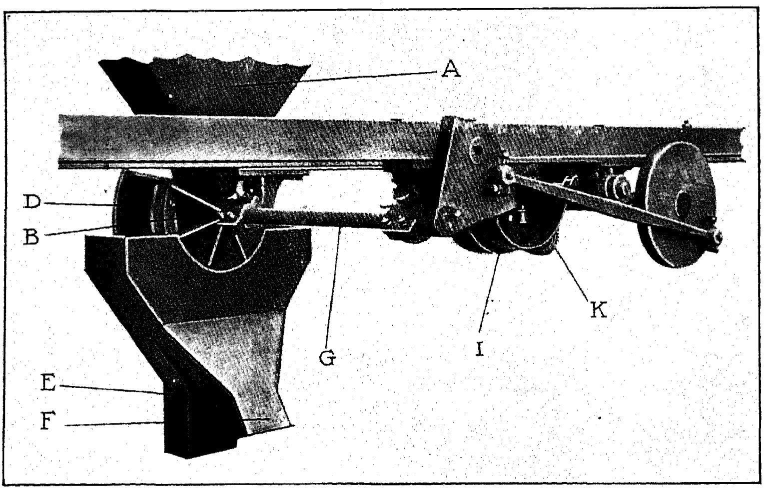


Fig. 14.—The Brunton Time-Sampler. Rear View.

drill-rags, which the mill attendants often endeavor to remove by pounding the sides of the spout while the machine is in motion, thereby distorting the form of the intake-spout very considerably from a true sector, and rendering it impossible to obtain a correct sample unless the delivery-stream is perfectly homogeneous, which is never the case. The great length of the radial edges of the sector intake-spout renders them, of course, peculiarly susceptible to be thrown out of alignment, and manufacturers of this class of machines should do something to shorten up the length of the radial edges, or stiffen them to prevent accidental distortion. At first sight it might be thought that this could be accomplished by reducing the size of the sector, but experience has shown that the width of any spout, delivery, or intake should be something more than three times the greatest diameter of the coarsest particle passing through it; otherwise, a bridging effect occurs which affects the flow and often chokes up the spout. It is, therefore, good practice to make the width of the feed- and intake-spouts four times the diameter of the coarsest particles passing through them.

The Brunton time-sampler oscillates in a vertical plane through an arc of 120° instead of revolving in a horizontal plane like the sector-intake machines, an arrangement which permits the use of a rectangular intake-spout with cutting-edges so short that accidental distortion is impossible, while the tilting of the cutters at the end of the swing materially assists in dislodging any rags or strings which may have fallen on the cutting-edges. This construction requires less head-room than any other system, which effects a great saving in the cost of mill-construction, since it not only reduces the necessary height of the building but shortens all spouts and conveyors. The design of this machine cannot be very clearly shown in a linear drawing, but may be readily understood from Fig. 13, which is a front view of the sampler, having the housing open for cleaning. The various parts are explained as follows: *A*, receiving-hopper from crusher or rolls; *B*, delivery-spout; *C*, sample-intake; *DD*, "reject" divisions; *E*, housing-spout leading to the sample-bin; *F*, reject-spout leading to the shipping-bins; *G*, oscillator-shaft; *H*, gear-shift; *I*, driving-pulley; *J*, spur-gear; *K*, eccentric gear. Ordinarily the machine is driven by the spur-gear, *J*, in which case a 20-per

cent. sample is taken, but when a 5-per cent. sample is required the gear is slipped along the shaft, disengaging the spur-gear and bringing the eccentric gear, *K*, into play. Another advantage in the use of this machine is that, as the discharge of the ore from the sampler is assisted by centrifugal force instead of being retarded thereby, as is the case with all sector machines, it can be run at a much higher rate of speed, thereby increasing the number of samples per minute. This arrangement insures greater accuracy, since the more samples which can be cut from the falling ribbon without "batting" the ore too vigorously with the sides of the cutters, the better are the chances for obtaining an exact average of the stream. Fig. 14 shows a rear view of the same machine and gives a clear idea of the driving mechanism and a study of the relations between the oscillator rocker-arm and the disk-crank.

While there seems to be a general impression among mining-men that high-grade ores are more difficult to sample correctly than those of low grade, there is no reason for this assumption. The difficulty of sampling accurately increases directly as the difference between the value of the highest- and the lowest-grade material contained in the lot, and is at its maximum when the values are carried in large masses of metallics or crystals of very rich minerals occurring in barren rock.

If we imagine a lot, for instance, of Cripple Creek ore, composed entirely of barren gangue and one solitary piece of calaverite, it would be manifestly impossible to sample such a lot of ore without crushing, since in any subdivision either the sample or the reject would contain all of the mineral.

Suppose this lot to be subjected to a slight crushing and the solitary piece of mineral broken into three fragments, then dividing the lot into halves would at the best throw 50 per cent. more value into the one half than into the other; it is therefore clearly manifest that in order to obtain a sample which shall correctly represent this or any other lot, it is necessary to crush it to such a degree of fineness that one particle more or less taken into the sample shall not materially affect its metallic content. In other words, the maximum error is determined by the ratio of the weight of the largest particle of metal or high-grade mineral to the weight of the entire lot. At this point another condition must be considered. In any lot

of ore it is easy to see that the chances of finding a full-sized piece of the highest-grade material would be much greater on

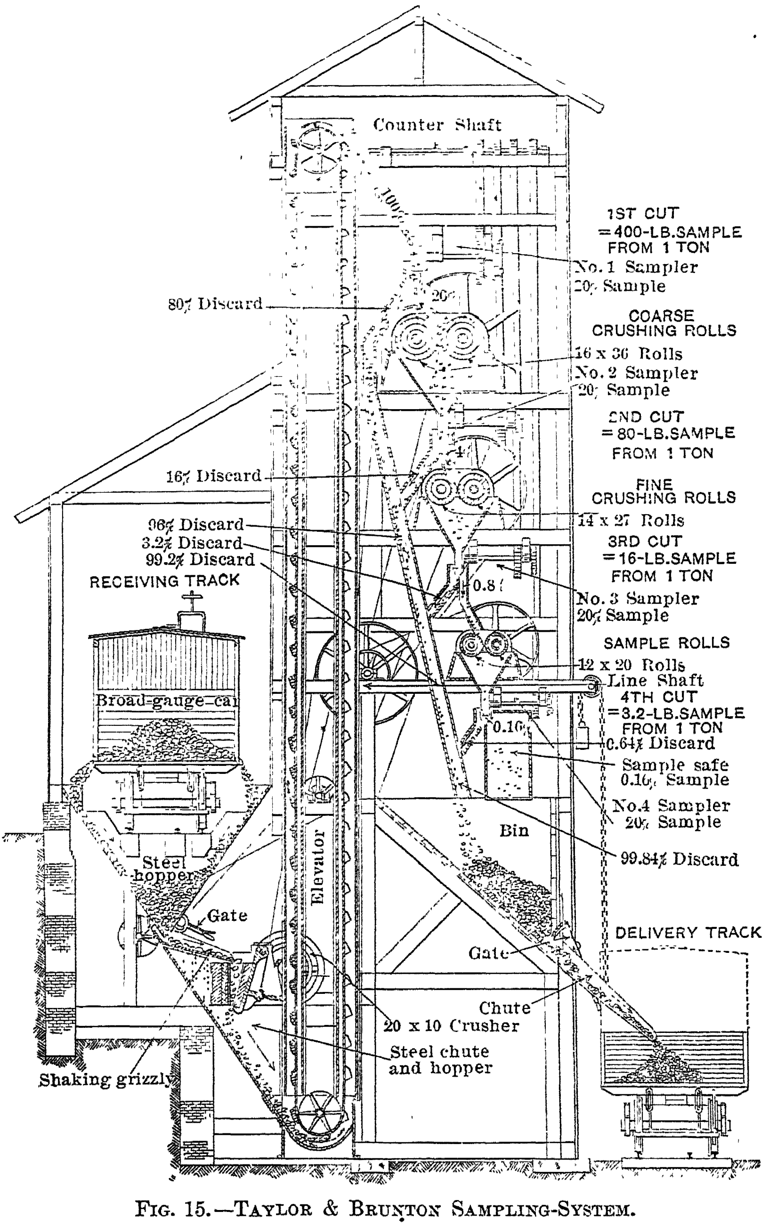


FIG. 15.—TAYLOR & BRUNTON SAMPLING-SYSTEM.

a lot of ore crushed to 0.25-in. cubes than in a lot crushed in 1-in. cubes, therefore accuracy demands that the ratio between

the weight of the largest particle and the entire lot shall increase directly as the fineness.

In this particular, practice and theory are in complete accord, and all of the latest and most improved mills practice alternate crushing and subdivision from the coarsest size down to the finest. It is customary at each successive stage to reduce the diameter of the coarsest particles one-half, thus decreasing

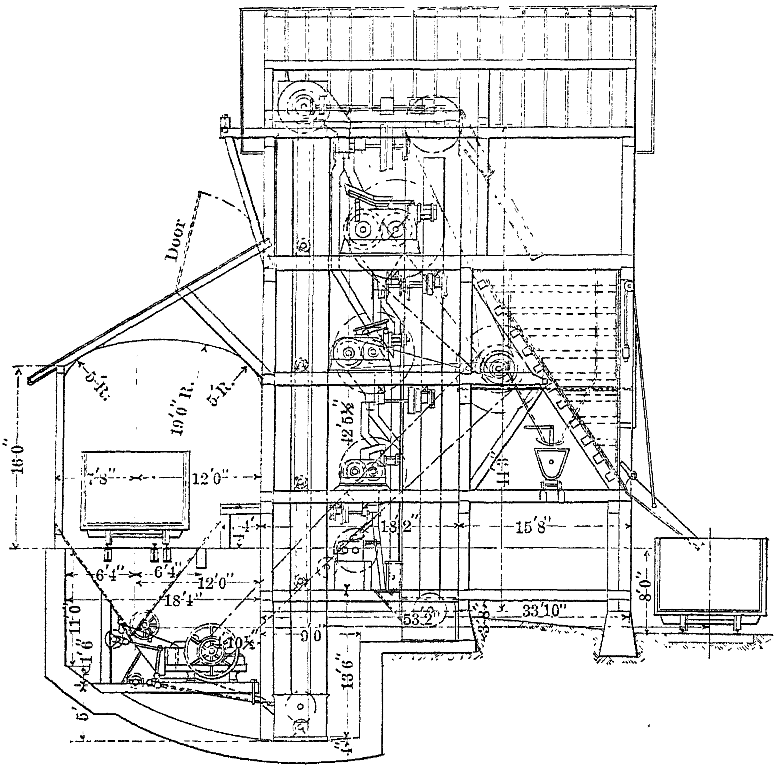


Fig. 16.—Vertical Section of the Taylor & Brunton Sampling-Mill, Silver City, Utah.

the volume to one-eighth, or 12.5 per cent. The usual sample taken at each successive stage is 20 per cent., so that while the size of the particle at each step has been reduced to 12.5 per cent., the amount of sample taken is 20 per cent., consequently the ratio between the weight of the largest particle and the weight of the sample rises steadily from the beginning of the series of operations to the end, thereby meeting the conditions theoretically necessary to an accurate determination of value.

An ideal sampling-mill, where the situation and nature of the service will permit this form of construction, is shown in Fig. 15. This plant is entirely automatic, and when the ore is received in hopper-bottom cars no manual handling is required at any stage, while the sample is automatically delivered into a

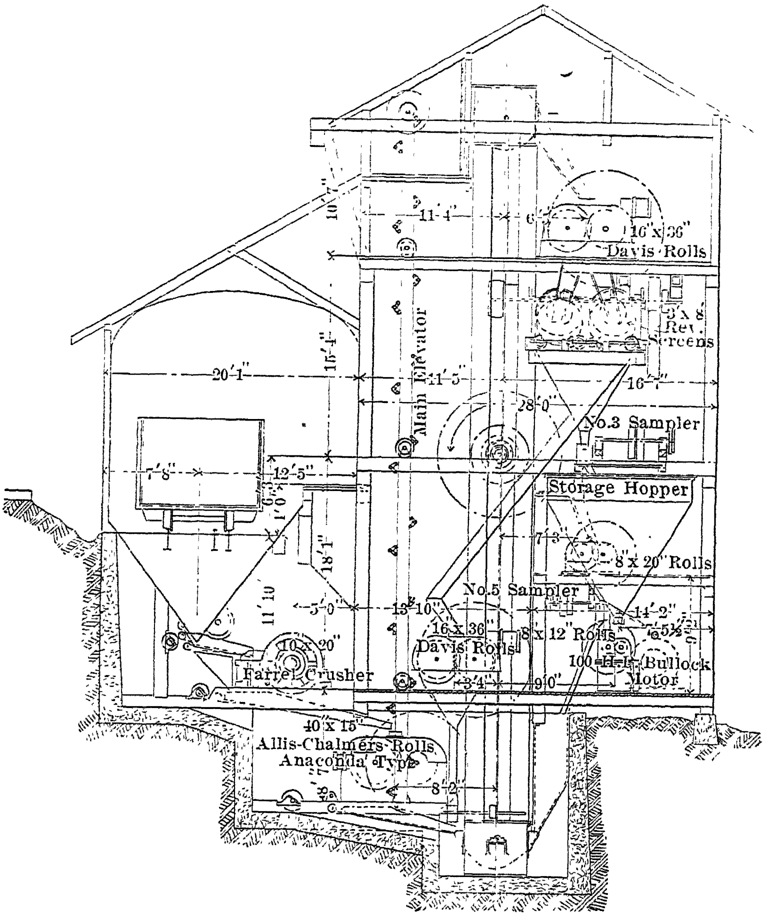


FIG. 17.—VERTICAL SECTION OF THE MATTE AND SULPHIDE SAMPLING-MILL OF THE TINTIC SMELTING CO., SILVER CITY, UTAH.

locked steel safe. To simplify the drawing, the roll-feeders have been omitted.

Fig. 16 is a vertical longitudinal section of the new Taylor & Brunton mill at Silver City, Utah, completed January, 1909. Like the plant shown in Fig. 15, it is automatic throughout,

electric driven, and contains every modern device for facilitating crushing, sampling, and cleaning, the latter operation being performed by compressed air.

TAYLOR & BRUNTON SAMPLING WORKS, SILVER CITY, UTAH.

Calculations based on 25 ton lot. Capacity 60 tons per hour

3 rail receiving tracks R.G.W., San Pedro and Eureka Hill Rys.

Steel ore hopper under railway tracks 14' 6" wide x 15' 0" long.

Shaking grizzly separating coarse and fines.

10" x 20" Farrel-Bacon crusher crushing to 2-1/2" cubes. 260 RPM.

Shaking tray elevator feeder 1-1/4" stroke. 250 RPM.

55' 0" vertical elevator belt 20", buckets 6" x 18". Speed 37.5 RPM.

No.2 Brunton 20% Sampler 7" C-C 19 RPM.

Reject 80% — 20% Sample = 10,000 lbs. = 1/3500-

Shaking tray roll feeder 3/4" stroke. 233 RPM.

16" x 36" Davis belted rolls crushing to 1" cubes. 50 RPM.

No.3 Brunton 20% Sampler 5-3/4" C-C. 20 RPM.

Reject 16% — 4% Sample= 2000 lbs.= 1/11,000.

Shaking tray roll feeder 5/8" stroke. 235 RPM.

15" x 27" Gates belted rolls crushing to 3/8" cubes. 68 RPM

No.4 Brunton 20% Sampler 4-1/2" C-C. 26 RPM.

Reject 3.2% — 0.8% Sample= 400 lbs.= 1/42,000.

Shaking tray roll feeder 1/2" stroke. 180 RPM.

8" x 20" Davis belted rolls crushing to 1/8" cubes. 80 RPM.

No.5 Brunton 20% Sampler 3-1/2" C-C- 33 RPM.

Reject 0.64% — 0.16% Sample= 80 lbs.= 1/240,000.

Shaking tray roll feeder 1/2" stroke. 185 RPM.

8" x 12" Davis finishing rolls crushing to 14 mesh. 100 RPM.

Locked steel sample safe.

Covered steel sample buggy.

Locked cutting room with steel floor and observation windows.

T.& B. precision 3/4" div. splitter to 10-12 lbs.=1/1,100,000.

Electric drier. Temperature 250° F.

Two Engelbach grinders to 50 mesh cone makers, 86 RPM.

1/2" division T.& B. precision splitter to 20-24 ozs.=1/7,000,000.

Bucking Board to 120 mesh.

Rubber rolling cloth five minutes.

1/4" div. T.& B. precision splitter to 4 sample sacks 5-6 ozs.each.

Loading out Bins.

Reject Elevator.

FIG. 18.—FLOW-SHEET, TAYLOR & BRUNTON SAMPLING WORKS, SILVER CITY, UTAH.

A good example of a modern crushing-, screening-, and sampling-plant is shown in Fig. 17, which is a longitudinal section through the new matte and sulphide mill of the Tintic Smelting Co. at Silver City, Utah.

It is not the purpose of this paper, however, to take up and illustrate details of construction, but rather to show the methods which are being employed to produce the best results in the

SYNOPSIS OF MACHINERY AND METHODS.

MATTE & SULPHIDE MILL, TINTIC SMELTING COMPANY:

Calculations based on 35 ton lot. Capacity 30 tons per hour.

3 rail tracks, R.G.W., San Pedro and Eureka Hill Rys.

Steel receiving hopper under railway tracks, 14' 6" wide x 15'0" long.

Shaking grizzly separating coarse and fines.

10" x 20" Farrel-Bacon crusher, 260 RPM, crushing to 2" cubes.

Shaking tray roll feeder 1-1/4" stroke, 250 R.P.M.

15"x 40" Allis-Chalmers, Anaconda Type, belted rolls, 45 RPM, crushing to 1" cubes.

Shaking tray elevator feeder 1-1/4" stroke. 250 R.P.M.

55' 9" C-C elevator belt 20", buckets 6"x 18". Speed 355' F.M.

16" x 36" Davis belted rolls, 50 RPM. , crushing to 1/2" cubes.

2 revolving screens each 30" dia. 8'0" long, 2-1/2 mesh, 1/4" openings.

Sized Product — Oversize.

No.3 Brunton 2% Sampler 15 RPM 2-5/16" C-C of cutting edges. — 16" x 36" Davis finishing rolls, 50 RPM, 1/4" cubes.

Reject 98% — 2% Sample — Elevator

Storage hopper — * 1000 lbs.=1/350,000 — Screens

Robbins belt

Sulphide beds

Shaking tray transfer feeder 1/2" stroke. 180 RPM.

8" x 20" Davis belted sample rolls crushing to 5 mesh. 75 RPM.

No.5 Brunton 10% Sampler, 31 RPM, 1-3/4" C-C.

Reject 1.8% — 0.2% Sample=100 lbs.= 1/440,000.

Auxiliary hopper and run to storage hopper during cleanup

Shaking tray roll feeder 1/2" stroke. 185 RPM.

8" x 12" Davis finishing rolls crushing to 14 mesh. 100 RPM.

Locked steel sample safe

Covered sample buggy.

Locked cutting room with steel floor and observation windows.

T.& B. precision splitter to 10-12 lbs.= 1/1,100,000.

Covered and locked steel bucket to grinding room in sampling mill.

FIG. 19.—FLOW-SHEET, TINTIC SMELTING CO., SILVER CITY, UTAH.

valuation of ores; an operation which means everything to the mining and metallurgical industries. One of the first requisites for successful mining is an accurate knowledge of what a

property is producing, and this of necessity involves correct sampling, both underground and on the surface. The first

SYNOPSIS OF MACHINERY & SAMPLING SYSTEM

AT THE WESTERN ORE PURCHASING CO'S. WORKS, M I L L E R S, NEVADA.

Computations based on 25 ton lot. Capacity 70 tons per hour.

On main line of Tonopah & Goldfield Railroad Co.

Four storage tracks. 6000 feet private track.

Receiving Hopper under railroad track.

Power 75 H.P. Westinghouse Motor. 710 R.P.M. 2200 Volts

Two No.4 Gates crushers, Style K, crushing to 2" 420 RPM

Elevator 71' 6" C-C. Buckets 16" x 6" x 6". Speed 420' P.M.

No.1 Special 20% C. Snyder Sampler, delivery spout 7"x35" 10 RPM.

Discard 80% 20% Sample = 10,000 lbs. = 1/7,000.

16" x 36" Davis belt-driven rolls, crushing to 1". 70 RPM.

Elevator 31' 3" C-C. Buckets 4" x 3". Speed 340' P.M.

No.1 Chas. Snyder 20% Sampler, delivery spout 5"x 25". 10 RPM.

Discard 16% 4% Sample = 2000 lbs. = 1/11,000.

14" x 30" Davis belt-driven rolls, crushing to 3/8". 110 RPM.

Loading Hopper

Elevator 28' 9" C-C. Buckets 3" x 5". Speed 340' P.M.

No.2 Chas. Snyder 20% Sampler, delivery spout 3"x 15". 10 RPM.

Discard 3.2% 0.8% Sample = 400 lbs. = 1/42,000.

12" x 20" Davis belt-driven rolls, crushing to 6 mesh. 110 RPM.

No.3 Chas. Snyder 20% Sampler, delivery spout 2" x 10" 10 RPM.

Discard 0.64% 0.16% Sample = 80 lbs. = 1/670,000.

R.R. Cars standing on 50' 100 ton Fairbanks reg. beam scales.

Sample Hopper car, bottom discharge.

Enclosed Sample Room. 10 H.P. Motor 710 R.P.M.

7" x 14" Stearns-Roger rolls, crushing to 8 mesh. 150 RPM.

3/4" div. riffle box to 50-150 lbs. = 1/1,040,000.

Regrind through closed 7" x 14" rolls, to 16 mesh.

Grinding Room.

1/2" div. riffle box to 12-15 lbs. = 1/1,600,000;

Electric drier. Temperature 240°- 380° F.

Two No.2 Englebach Grinders to 50 mesh. Speed 60-130 RPM.

Rubber rolling cloth three minutes.

1/2" div. riffle box to 20-25 ounces = 1/7,000,000.

Grinder and Bucking Board to 120 mesh.

Rubber rolling cloth 10 minutes.

1/4" div. riffle to 4 sample sacks 5.5 ounces each

Reject Elevator.

FIG. 20.—FLOW-SHEET, WESTERN ORE PURCHASING CO., MILLERS, NEV.

essential to success in all metallurgical work is a knowledge of the exact value of the ore going into the works, and of the different products issuing therefrom.

In order to show the methods of operation in vogue in different districts, I present Figs. 18, 19, 20, and 21, which contain the flow-sheets of a number of the newest and largest sam-

SYNOPSIS OF MACHINERY AND METHODS AT
THE AMERICAN S. & R. COMPANY SAMPLER No. 2, MURRAY, UTAH.
Computations based on 25 ton lot. Capacity 50 tons per hour.

Rio Grande Western & S. P. L. A. & S. L. Railways.
↓
Stub end elevated receiving track 10' above mill floor.
↓
Wide chute at level car floor, feeding direct to crusher.
↓
Gates crusher #4 style D, crushing to 3".
↓
Ninety foot elevator, buckets 6" x 8" x 12".
↓
Vezin 20% 20" radius double spout sampler. 13 R.P.M.
Reject 80% → Loading bins over railroad track. | 20% sample = 10,000 lb. = 1/2000.
↓
Gates crusher #2 style H, crushing to 1-1/4".
↓
Vezin 20% 20" radius sampler.
Reject 16% → Loading bins over railroad track. | 4% sample = 2,000 lb. = 1/5000.
↓
15" x 26" Gates belted rolls, crushing to 3/4".
↓
70 foot elevator 4" x 4" x 6" buckets.
↓
Vezin double spout 20% sampler.
Reject 3.2% → Auxiliary ore bin. | 0.8% sample = 400 lb. = 1/5300.
↓
12" x 20" high-speed Allis Chalmers rolls, crushing to 3/8"
↓
Vezin double spout 20% sampler.
Reject 0.64% → Auxiliary ore bin. | 0.16% sample = 80 lb. = 1/8500.
↓
Steel safe in grinding room.
↓
Entire sample carried over to #3 mill.
↓
Run through 12" x 20" rolls, crushing to 3/16".
↓
3/4" Rhodes splitter to 75 lb. = 1/62500.
↓
Surface dryer in #4 mill grinding room.
↓
Two Engleback grinders to 50 mesh.
↓
1/4" riffle to 48-50 ounces = 1/16,000,000.
↓
Bucking board to 100 mesh Ag. and 120 mesh Au.
↓
Gold pan mixing five minutes.
↓
1/4" riffle to four 8 oz. pulp samples.

FIG. 21.—FLOW-SHEET, AMERICAN SMELTING & REFINING CO., SAMPLER NO. 2, MURRAY, UTAH.

pling-works, clearly showing every detail of the process, and the machinery employed in the alternate operations of crushing and subdivision, as well as the increase of ratio as the final stages are reached. This style of flow-sheet was originally

type-written on ordinary 8.5- by 13-in. paper. perforated for a loose-leaf binder. In this way flow-sheets of many classes of operations may be preserved in convenient form.

These flow-sheets show considerable differences at all stages, and a great divergency in the methods of subdividing the final sample. Too many manual operations are in use, and there is no doubt that the complete elimination of the personal equation by using a small Taylor & Brunton splitter with $\frac{3}{8}$-in. riffles (shown in Fig. 12) gives by far the most accurate subdivision.

To show how closely results between different mills and repeat-sampling in individual mills may be made to check, the following examples, taken at random, should suffice:

TABLE I.—*Sampling-Results, Taylor & Brunton Sampling Co., Cripple Creek, Colo.*

Lot No.	Sample.	Resample.
	Gold.	Gold.
	Ounces per Ton.	Ounces per Ton.
3192	3.62	3.64
3198	5 04	5.015
3219	2.70	2.67
3235	3.18	3.16
3310	1.17	1.17
3324	6.52	6.51
3340	0.71	0.78
3388	1 70	1.84
3424	9.24	9.20
3471	30.64	30.52

TABLE II.—*Sampling-Results, Taylor & Brunton Sampling Co., Cripple Creek, Colo.*

Lot No.	Mine.	First Sample.		Resample.		Settlement.
		Gold.		Gold.		
		Mill-Assay.	Mine-Assay.	Mill-Assay.	Mine-Assay.	
		Oz.per Ton.	Oz.per Ton.	Oz.per Ton.	Oz.per Ton.	Oz.per Ton.
4514	Sacramento..........	2.22	2.24	2.22	2.23	2.225
4604	Little Clara.........	115.05	115 25	114 90	115.20	115.03
4705	Mary Cashen.........	1.11	1.10	1.07	1 09	1.08
4726	Midget................	1.27	1.30	1.30	1.35	1.325
4853	Independence, Ltd.	1.36	1.35	1 29	1.30	1.295
4914	Bon. King...........	0.53	0.55	0.55	0.56	0.555
5062	Little Clara..........	1.77	1.72	1.75	1.74	1.745
5272	Old Abe...............	1.27	1.24	1.27	1.28	1.27
5753	Independence, Ltd	2.33	2.34	2 34	2.36	2.35
5913	Little Clara.....	12.62	12.58	12.69	12 68	12.695

TABLE III.—*Sampling-Results, Taylor & Brunton Sampling Co., Cripple Creek, Colo.*

Lot No. of Mixture.	Original Purchase.		Mixture.	
	Weight.	Gold-Assay.	Mathematical Average.	Mechanical Sample.
	Pounds.	Ounces per Ton.	Ounces per Ton.	Ounces per Ton.
5394	17,588 9,646 11,348	0.98 1.17 0.875	0.996	1.00
5496	17,405 6,615 17,123	0.98 0.895 0.995	0.972	0.975
5799	422 12,851 175 21,278	8.24 2.225 8.50 1.85	2.099	2.14
5890	19,090 8,761 8,852	1.925 1.97 1.89	1.927	1.93
3465	5,274 17,935	2.10 1.89	1.937	1.97
3678	3,795 17,122 11,357 6,592	1.88 1.49 1.345 1.465	1.481	1.52
3850	3,633 16,803 8,360 11,222 3,731	3.365 4.675 5.82 3.73 36.445	7.252	7.24
4170	18,605 18,621 11,937 8,593	0.83 0.77 1.42 0.98	0.954	0.92
4292	17,848 15,435 17,436	1.165 0.615 1.12	0.982	0.96
4319	4,014 15,611 13,334 11,712	2.835 2.24 3.35 2.58	2.71	2.75

TABLE IV.—*Sampling-Results, American Smelting & Refining Co., No. 2 Sampling-Mill, Utah.*

Using Vezin Samplers.

Number.	Size of Lots. Tons Dry.	First Sample.		Resample.	
		Gold.	Silver.	Gold.	Silver.
		Oz. per Ton.	Oz. per Ton.	Oz. per Ton.	Oz. per Ton.
1	131	5.18	1.1	5.02	1.1
2	138	4.67	trace	4.82	trace
3	85	2.45	1.0	2.45	1.0
4	75	3.49	5.3	3 45	5.5
5	104	2.48	1 0	2.41	1.0
6	138	2 31	trace	2 39	trace
7	97	2.43	2.0	2.31	2.0
8	96	2.43	1.4	2.38	1.2
9	83	2.47	1 5	2.48	1.7
10	91	5.08	trace	4.94	trace
Average.....	103.8	3.299	1.33	3.265	1.35

TABLE V.—*Sampling-Results, Western Ore Purchasing Co. Plants.*

Using Charles Snyder Samplers.

Miller's Plant:

	First Sample. Gold. Ounces Per Ton.	First Sample. Silver. Ounces Per Ton.	Resample. Gold. Ounces Per Ton.	Resample. Silver. Ounces Per Ton.
Lot No. 4979, Assayer A, ...	0.21	36.45	0.21	36.35
Assayer B, ...	0.20	36.35	0.20	36.85
Average, ...	0.205	36.40	0.205	36.60

Columbia Plant:

	First Sample. Gold. Ounces Per Ton.	Resample. Gold. Ounces Per Ton.
Lot No. 844, average of two assays, ...	5.393	5.37

Hazen Plant:

	First Sample. Gold. Ounces Per Ton.	First Sample. Silver. Ounces Per Ton.	Resample. Gold. Ounces Per Ton.	Resample. Silver. Ounces Per Ton.
Lot No. 1131, ...	1.76	4.50	1.743	4.65

TABLE VI.—*Sampling-Results, Columbia Plant.*

Lot Mixture No. 473.

Lot Number.	Dry Weight.	Assay Gold.	Assay Silver.	Gold-Content.	Silver-Content.
	Pounds.	Oz. per Ton.	Oz. per Ton.	Ounces.	Ounces.
972	78,884	1.91	1.10	75.33	43.38
961	78,408	1.82	0.90	71.35	35.28
974	78,837	1.69	0.80	66.62	31.53
979	37,352	4.23	......	79.00	....
1145	7,119	0.30	161.40	1.07	574.50
	280,600	....	....	293.47	684.69
Mathematical average.....		2.09	4.89		
Actual sample of mixture:					
	280,364	2.07	4.83	290.17	676.29

Table VII. gives a comparison on a lot of Bullfrog Pioneer ore sampled at Columbia plant, and afterwards screened through a ⅜-in. screen at Hazen; fines sold to reverberatory and coarse to blast-furnace smelters, actual weights and moistures having been determined both on the fines and the coarse, which makes a showing of a slight loss in weights.

TABLE VII.—*Sampling-Results, Columbia Plant.*

Lot No. 1017.	Dry Weight.	Assay Gold.	Total Gold-Content.
	Pounds.	Ounces per Ton.	Ounces.
	122,189	3.71	226.66
		After screening:	
Fines...........	36,909	6.06	111.83
Coarse.........	84,760	2.75	116.55
	121,669		228.38

Table VIII. gives a comparison of assays and total ounces of gold contained in four lots of Engineers' Lease ore from the property of the Florence-Goldfield Mining Co., in Goldfield, Nev., sampled at the Columbia plant and afterwards screened through ⅜-in. punched screen at the Hazen plant, and the coarse and fines sampled separately after screening.

The dry weights show the same in each case, due to the fact that the fines after screening at Hazen were actually weighed and the moisture determined, thus ascertaining the exact dry weight, which was deducted from the total purchased dry weight, making a figured dry weight of the coarse.

TABLE VIII.—*Sampling-Results, Hazen Plant.*

Lot No.	Dry Weight of Ore.	Assay Gold.	Total Gold-Content.
	Pounds.	Ounces per Ton.	Ounces.
861	70,636	7.26	256.41
872	72,682	7.45	270.74
	143,318		527.15
		After screening:	
Fines.........	57,425	7.12	204.43
Coarse.......	85,893	7.42	318.66
	143,318		523.09
829	79,916	8.92	356.43
834	81,210	8.91	361.79
	161,126		718.22
		After screening:	
Fines.........	58,396	9.83	287.02
Coarse.......	102,730	8.44	433.52
	161,126		720.54

TABLE IX.—*Sampling-Results, Copeland Sampling Co., Victor, Colo.*

Using Oscillating Time-Samplers.

Mill Mixes on Cripple Creek Gold-Ore:

Lot No.	Weight. Pounds.	Assay Gold. Ounces per Ton.	Gold. Ounces per Ton.
603	2,237	17.81	
	1,223	25.685	
	1,705	67.07	
	5,183	1.25	
	6,846	2.59	
	10,015	0.485	
	18,488	1.545	
Mathematical average,		5.322	Machine-sample of mix, 5.35
907	1,759	1.795	
	13,220	2.54	
	19,271	1.72	
Mathematical average,		2.04	Machine-sample of mix, 2.04
941	16,696	1.28	
	17,179	0.79	
	15,066	1.50	
	2,729	1.39	
Mathematical average,		1.187	Machine-sample of mix, 1.23

Lot No.	Weight. Pounds.	Assay Gold. Ounces per Ton.	Gold. Ounces per Ton.
976	7,645	2.80	
	11,117	1.97	
	2,828	6.69	
	2,899	4.925	
Mathematical average,		3.124	Machine-sample of mix, 3.12
669	18,005	1.83	
	22,534	1.48	
Mathematical average,		1.07	Machine-sample of mix, 1.62
791	8,254	4.93	
	10,130	2.38	
	8,346	2.08	
Mathematical average,		3.073	Machine-sample of mix, 3.12

TABLE X.—*Sampling-Results, Copeland Sampling Co., Victor, Colo.*

Using Oscillating Time-Samplers.

Cripple Creek Gold-Ore:

Lot No.	First Sample. Gold. Ounces per Ton.	Resample. Gold. Ounces per Ton.
260	14.065	13.96
270	1.01	0.99
606	0.56	0.54
639	0.59	0.60
692	1.28	1.30
707	1.30	1.25

The most convincing tests of correct valuation in ore-sampling are those in which numbers of small lots are bought and paid for individually, and stored for a considerable time, until a sufficient quantity of ore has been collected to form one large lot. When this period arrives the individual lots are not mixed, but run through the mill in succession, and it is usually found that the mechanical sample of the mixture agrees with the calculated average, as determined by the values in the original purchases, as closely as the best control-assays.

The small lots when originally received, sampled, and purchased were coarse and generally wet, but when run through the mill the second time they are both fine and dry, giving thereby the greatest possible dissimilarity in conditions of size of particles and moisture-content. The excellent checks ob-

tained on this class of work show conclusively that with "time-sampling" the results obtained are in no way affected by the physical conditions of the ore, and may be implicitly accepted as correct.

The art of sampling has now reached a stage where a standardization of methods is both desirable and possible, and it is to be hoped that the Mining Congress, or the proposed Bureau of Mines, will take the matter under consideration and appoint a thoroughly qualified commission which will give the subject the study and investigation its importance demands. Recommendations by an unbiased, competent board would do much to eliminate faulty methods, and bring about the adoption of standard systems of valuation, which would prove of inestimable benefit to the mining and metallurgical industries from both a business and a scientific stand-point.

The Conservation of Coal in the United States.*

BY EDWARD W. PARKER, WASHINGTON, D. C.

(Spokane Meeting, September, 1909.)

If one is to place any credence at all in the reports published in the daily press, the subject of conservation has been a very lively topic of conversation during the past 60 days, and it does not appear that thė temperature of the summer months has been in any way moderated by the discussion. It is a subject in which we are all vitally interested and to which, so far as our mineral resources are concerned, both the Institute and the Geological Survey have liberally contributed. The report of the National Conservation Commission, appointed by President Roosevelt, contains a series of papers on the conservation of mineral resources, all of which were prepared by members of the Geological Survey and were compiled largely from information previously collected by that Federal bureau in the performance of its regular duties. It is not the purpose of this paper to reiterate *in extenso* any of the material already published. The contributions of the Survey officials to the Com-

* By permission of the Director of the U. S. Geological Survey.

mission's report have been published in a separate document as *Bulletin No.* 394. This document is for free distribution and may be obtained upon application to the Director of the Survey. In the preparation of this paper I desire merely to make a few suggestions regarding the possible necessity of some restraint upon or control of one branch of the mining industry with which I have been somewhat closely associated for the past 20 years—that of coal.

Most of the members of the Institute are cognizant of the suits brought by the government against the anthracite-operators in Pennsylvania, or the combination of interests commonly known as the "hard-coal trust." No defense of any illegal combination in restraint of trade is intended, but there are some facts which should not be lost sight of, and unfortunately those whose opinions are based upon the "news" given to us by the daily press are likely to be governed by *ex parte* testimony. The present situation in the anthracite-region is one that has been developed through sheer necessity, if the conservation of the supply of anthracite and the prolongation of the life of the fields in the best interests of the people were to be attained in any other way than through government control, and government control did not seem to be materializing. I believe that even Doctor Raymond will subscribe to the statement that a good part of the history of anthracite-mining has been one of profligate waste in the mining, preparation, and use of that precious supply of fuel; and this has only been remedied, none too soon, and could, under the circumstances, only be remedied, by the close control and conservative management which have been brought about in recent years. And I might pause here to pay a merited tribute to such men as Doctor Raymond, Eckley B. Coxe, P. W. Sheafer, Franklin B. Gowen, William Griffith, and a few others through whose efforts many reforms which lessened the waste of anthracite were effected. They were the pioneers in the battle for conservation, and a monument should be erected to them.

The securing by the Reading R. R. for its offspring, the Philadelphia & Reading Coal & Iron Co., of the great coal-reserves it owns to-day, was the beginning of a great movement which was foreseen by those in a position to see. The Reading company was temporarily bankrupted through its guarantee of the

debt thus incurred, but the possession and control of those coal-lands are indirectly the most valuable assets of the railroad at the present time. More than this, however, in the ultimate economy of things, has been the preservation of thousands of acres of coal-lands from reckless spoliation. The way was paved for the safe and sane control of the anthracite industry, albeit by a trust, and a stop was put to the cut-throat competition and extravagant methods which in earlier years had resulted in losses of millions of dollars in money and more than millions of tons of coal.

Under former conditions in the anthracite-regions, when it was not considered necessary to give thought to the morrow, and indeed up to the time when the Anthracite Coal Waste Commission made its report in 1887, it was estimated that for every ton of coal mined and sold, 1.5 tons were lost. The greater part of this loss was in the coal left in the ground as pillars to protect the workings, while millions of tons of small coal or screenings were thrown on the culm-banks which now form unsightly mountains in the coal-regions. Improved methods of mining and of preparation have of late years reduced the percentage of waste, so that at present the recovery will average about 60 per cent. and the loss about 40 per cent. By the means of washeries, usable coal is being saved from the old culm-banks, and specially-designed furnaces have made it possible to use this fuel in steam-plants. It may also be possible in the future to recover a considerable part of the coal from the pillars in the old workings where they have not been hopelessly crushed by the settling of the overlying strata; but this could be done only at enormous expense compared with the present mining-cost, and when the burning of anthracite coal shall have become a luxury, permitted only to the wealthy. Even in our day and generation it is only by the strictest economy and skillful management in the operation of the mines that the price of coal to the consumer can be preserved as at present. The average price of anthracite at the mine ranges from $2.25 to $2.35 per long ton. What are known as "prepared sizes"—lump, broken, furnace, egg, stove, and chestnut—range from $3 to $3.75, and all the profit must be made on these. Pea and smaller sizes are sold at less than the cost of production, some as low as from 40 to 50 cents a

ton. A careful study of conditions in the anthracite-region will convince the most skeptical that no robbery of the public is now being carried on.

The securing of the close control or practical monopoly that exists in the anthracite-region of Pennsylvania has been made possible by the comparatively limited area of the fields. The total area is less than 500 sq. miles and conditions are ideal for a natural monopoly. It is different in the bituminous fields, which are scattered over 30 different States and Territories. These fields aggregate about 250,000 sq. miles in area, exclusive of approximately equal areas of lower-grade coals and lignites, and are for the most part easy of access, and do not require a very large amount of capital to develop a mine. A few thousand dollars is all that is needed at first, and as there are no restrictions on the development of new properties, every one owning land underlain by workable coal seems impelled to get it out, whether there is a profitable market for it or not. The United States produced in 1907, the banner year of industrial activity in the United States, nearly 400,000,000 tons of bituminous coal, including about 8,250,000 tons of sub-bituminous coal and lignite. In 1908, in spite of the business depression, the production was 332,500,000 tons. We could produce from the mines already open 600,000,000 tons, and we would not have to operate the mines on Sundays and holidays to do it. If the railroads supply the cars and the motive-power the mines will supply the coal. During 1907 there was, until the panic started in October, a widespread demand for transportation facilities which the railroad companies were unable to furnish. Complaints of car-shortage came from practically every important coal-producing district. The transportation interests were subjected to all sorts of condemnation for failure to serve their patrons and the public. While I am by no means a defender of the railroad companies (particularly since the Hepburn bill went into effect), it is fair to say that the inadequacy of the car-supply was in reality beneficial. It is doubtful if the markets could have absorbed, even in the phenomenal activity of 1907, an additional 5,000,000 tons of bituminous coal, only 1.25 per cent. more than the production. An increase of 5 per cent. in production and car-supply would unquestionably have created a surplus and resulted in a

general demoralization in values. Notwithstanding these conditions, which are fairly well known among the coal-men, new mining companies are constantly being formed and new properties opened up. The railroad companies are called upon to furnish additional switches and spurs and to provide more cars or to spread out even more thinly an already insufficient supply. As common carriers these companies cannot discriminate, and when called upon must furnish the transportation without favor. Each new mine thus opened calls for miners to work it, and miners, who are as a class nomadic, are inclined to seek employment in the newer mines. This condition reduces the supply of labor and curtails the productive capacity of the older mines; and as reduction of output means increased cost in operation, it would appear that many of these must close down as unprofitable before all of the coal that should be extracted is won.

Under our system of government and of control over mining-operations any effective way of curbing the tendency on the part of coal-land owners to develop their properties or of protecting capital already invested in the industry is not apparent. The spirit of rivalry that exists throughout the coal-producing regions, district competing against district, and State against State, makes it useless to hope that State legislatures will place any restrictions upon the industry which will discourage further development. Yet every new mine opened has its influence on the creation of a surplus, which, while it may seem desirable to those who clamor for cheaper coal, is ultimately destructive of industry, lowers wages, and makes necessary the practice of economies (?) that are prejudicial to safety to life and property in the operation of the mines.

The year 1907, if not the most prosperous year in the history of bituminous-coal mining, was one of the most prosperous years. Production reached its maximum, and prices were the highest in recent years. Yet there were very few districts in which the margin between the cost of putting the coal on the railroad-cars and the price at which it was sold was as much as 10 cents a ton. In many States it was considerably less than 10 cents, and this margin must cover such losses as are due to explosions and other accidents, indemnities paid to employees or their heirs, and all extraordinary expenses. One such explosion as

that at Monongah, W. Va., in December, 1907, will wipe out many years' profits. In 1908 not only was the margin of profit much reduced in all the coal-mining districts, but thousands and hundreds of thousands of tons were sold at less than the cost of production. Of course, it is poor business to continue production at a loss, but a coal-mine is not a factory or a quarry. To close down a coal-mine costs money. The mine must be kept clear of water; if the ventilation is stopped, gas accumulates; falls of roof and coal occur; and after a period of idleness much repair-work has to be done before operations can be resumed. It is often less expensive in the long run to continue the production of coal at a loss than to close down the mine.

It is, perhaps, somewhat bold to suggest that the bituminous mines should be put under some sort of government control; but if they are not, I am frankly of the opinion that before many decades have passed the protection of capital already invested will make it necessary to secure control by private enterprise of certainly the areas containing the higher grades of coal, and to regulate the production according to market requirements. Under our system of government the Federal authorities have no jurisdiction over mines in the several States, unless the power given them under the Constitution to regulate commerce between the States could be stretched to apply to coal because of its bearing on interstate traffic. But it does look as if a choice will have to be made from three evils. The first of these is the continuation of the conditions as they now exist—a feasting for to-day and remorse for the morrow. The second is the ultimate control by a combination of interests that will make the "hard-coal trust" appear insignificant—to look "like thirty cents," as expressed in the vernacular, and the "water-power trust" would be of still less importance. The third is governmental supervision and regulation—not ownership, however. The first will be bad, the second worse; the third is problematical. Under such government control bituminous-coal mining could be regulated through a system of license; and in order that restriction on coal-production may be secured, no license should issue for the opening of a new mine until ample proof is shown that the necessities of the people or of trade require it.

I do not believe that present conditions should continue—

they certainly must not continue if our coal-supplies are going to be considered—nor do I believe that it is the part of wisdom to permit the bituminous coal-supplies to get into the control of a comparatively few men living in New York and Chicago. It may be suggested that control by the several States is a fourth and best alternative. Under the competitive conditions to which I have referred it is not to be hoped or expected that the States will undertake to restrict developments in their respective jurisdictions any more than that they will enact legislation which will restrict the miner in his personal liberty.

And speaking of the personal liberty of the miner, it is well known that not the least difficulty experienced in carrying on a coal-mining operation is the enforcement of discipline among the employees.

When humanity is shocked by the occurrence of some great disaster in a coal-mine, sympathy is poured out to the miners and invectives hurled against the mine-owners. He is without a soul who would withhold sympathy at such a time, but scarcely less brutal is he who holds up to the condemnation of the world the ones in authority who have by all human endeavor striven to prevent the catastrophe. It is unfortunately true that the death-record in the coal-mines of the United States shows unfavorable comparison with other countries, but it cannot be truly said that the blame should attach to the operators alone. In the great majority of cases they who suffer death or injury in the coal-mines are victims of their own carelessness, or that of their fellow-employees. The year 1907, the one of greatest production in our history, was the darkest in regard to casualties, the death-list exceeding 3,000. At one time an epidemic of explosions seemed to exist, and scarcely had the echoes of one died away before another occurred. The victims from this cause—that is, from explosions alone—numbered nearly 1,000, or approximately one-third of the total number of men killed. The statistics show, however, that more than that number were killed by falls of roof, most of which are preventable if proper precautions are taken by the men, or if, in fact, they obey the rules of the companies. In ordinary years the majority of accidents are due to roof-falls or to other preventable causes, but these occur singly and are not chronicled in the news dispatches. Even in the case of explosions,

the cause may usually be traced, if any witnesses are alive to testify, to an act of carelessness or disobedience.

A prolific cause of mine-explosions is what is known as a "windy shot," due to an improperly-prepared blast, or to the failure on the part of the miner to undercut his coal, depending, as he frequently does, on the powder to do his work for him. Here it is that the strength of the mine-workers' union might be exercised for good, but unfortunately, instead of helping to secure legislation which will hold miners criminally responsible for acts of carelessness or insubordination that may result in loss of life or damage to property, the miners' influence is exerted against it. Such legislation means a restriction of their liberties as American citizens. If in the effort to enforce discipline a mine-employee is discharged for infraction of rules, the result is, in the majority of cases, the precipitation of a strike, and the mine is laid idle for several days at least. Coal-mining is at best a hazardous occupation, and there is no line of industry in which military discipline is so essential, except perhaps in the passenger service of railroads and steamships. In European countries, where fewer accidents occur, the operations are under strict police surveillance. Both miners and operators are made to obey the laws. When this is done in the United States accidents will decrease, but the expense of mining will be increased and the price of coal will advance. On behalf of the mine-owners, it should be admitted that self-interest, if nothing else, compels the exercise of precautions against accidents. If they have no interest in securing the safety of their employees, they have at least a desire to protect their own properties. There are instances, it is true, where operators, like the men, take chances, where false economies are practiced, and where even ordinary precautions are not observed, but these, I honestly believe, are rare, very rare, exceptions.

*

The Influence of Bismuth on Wire-Bar Copper.*

BY H. N. LAWRIE, PORTLAND, ORE.

(Spokane Meeting, September, 1909.)

Introduction.

THIS study was undertaken on account of the lack of definite knowledge concerning the influence of bismuth on wire-bar copper, and the small elimination of bismuth from copper-matte during the smelting-operations.

The early workers who studied the influence of bismuth on copper confined their investigations to malleability and ductility—two physical properties which are related to the others so intimately that their determination is of primary impórtance. Karsten [1] and Levol,[2] using bending- and mealleability-tests, seem to have been the first to take up the subject, and while their results are incomplete, they agree in the main with the later work of Hampe.[3]

The bending-tests used by the first investigators consisted in bending the specimen until it failed. The malleability-tests were made by beating the material to a knife-edge with a hammer. Hampe not only determined the percentage of bismuth which would cause red-shortness and cold-shortness of the alloy, together with other influences of bismuth on the malleability and ductility of copper, but also took up the influence of the bismuth protoxide, BiO, on copper, and of bismuth sesquioxide, Bi_2O_3, on the copper protoxide, CuO. Hampe's conclusions of the influences of these two latter compounds of bismuth on copper are as follows:

"If the protoxide of bismuth be alloyed with metallic copper, it is not changed to metallic bismuth, but remains mechanically distributed throughout the copper.

---

* Submitted in part fulfillment of the requirements for the degree of Engineer of Mines, to the Faculty of Columbia University in 1905, and accepted for publication in the *Transactions* of the American Institute of Mining Engineers.

[1] *Zeitschrift für das Berg- Hütten- und Salinen-Wesen im Preussischen Staate*, vol. xxii., pp. 93 to 138 (1874).

[2] *Ibid.*

[3] *Ibid.*

In this state the effect of bismuth seems to be less disadvantageous than in the metallic condition. The difference, however, is insignificant and restricted only to ductility, and then when cold. A material diminution in cold-shortness is shown, if the oxide of bismuth is combined with the protoxide of copper. Such alloys are far less cold-short than those containing the same amount of bismuth in the metallic form."

The Alloys Research Committee found a drop in tensile strength of 11,000 lb. per sq. in. in comparing two copper-alloys: one containing 0.10 and the other 0.20 per cent. of bismuth.[4] This result was the reason for including in their work the determination of the melting-point of the alloys. It was beyond comprehension why bismuth, which occurs in the Periodic Law next to arsenic and antimony, should diminish the tensile strength, while both arsenic and antimony have a tendency to increase it.

In order to study this strange influence of bismuth, Arnold and Jefferson[5] took up the subject from the stand-point of microstructure, and their conclusions corroborate the results of the Alloys Research Committee in their melting-point determinations. The results obtained by E. A. Lewis[6] agree with those of Hampe on malleability, and of Arnold and Jefferson on the structure of the alloy. Moreover, Lewis ascertained the influence of bismuth on copper, which contained also arsenic, tin, manganese, and aluminum individually. His conclusion is that "the injurious influence of bismuth is offset by the presence of arsenic, and intensified by tin, manganese, and aluminum."

Inasmuch as but little work has been done on tensile tests of copper-bismuth alloys, I decided to test the malleability and tensile strength of a series of copper-bismuth alloys, cast into bars, supplementing the work with similar tests on the same specimens passed through rolls, in order to determine the effect of rolling.

Recently, A. H. Hiorns[7] has published the conclusions of his

---

[4] *Proceedings of the Institution of Mechanical Engineers* (London), Part 2, p. 120 (Apr., 1893).

[5] *Engineering*, vol. lxi., p. 176 (Feb. 7, 1896).

[6] *Journal of the Society of Chemical Industry*, vol. xxii., No. 24, p. 1351 (Dec. 31, 1903).

[7] *Idem*, vol. xxiv., No. 9, p. 501 (May 15, 1905).

comprehensive investigation of the microstructure of copper-bismuth alloys.

With regard to the elimination of bismuth from copper-matte, Edward Keller[8] found that in refining the matte 54 per cent. of the bismuth was eliminated in the reverberatory furnace, while 95 per cent. was eliminated in the converter. Even though the converter process be used, the percentage of bismuth remaining in the refined copper is large, since a 95-per cent. elimination is lower than the extraction of arsenic, antimony, and sulphur, which is 98 per cent. or more.

Raw Materials.

The metals used in my investigation were purest wire-bar copper, obtained in ingot form from the Nichols Chemical Co., Laurel Hill, Brooklyn, N. Y., and metallic bismuth, c.p., supplied by Eimer & Amend. The bismuth was added directly to the molten copper instead of diffusing it in the main mass by first alloying it with a small amount of copper, as had formerly been the practice. The pure elements of the alloy were used in preference to copper of known bismuth-content for two main reasons: (1) the difficulty of procuring samples containing the exact percentage of bismuth best adapted to the research; and (2) the probable presence of other impurities which would interfere with the tests. As a check, a careful chemical analysis was made of one of the series prepared synthetically. The alloy, prepared to contain 0.20 per cent. of bismuth, gave on analysis 0.18 per cent. In the alloys containing a smaller proportion of bismuth the error was probably less than 10 per cent.

Casting the Test-Bars.

For the first melts on pure copper the ingot of wire-bar copper was cut by a hack-saw into cakes, each weighing about 3.5 lb. The cakes were melted individually in a No. 6 Dixon graphite crucible, previously heated to a bright red.

The melt, made in a No. 2 melting-pot of the American Gas Furnace Co., took from 20 to 30 min. During the melt the mold was heated over a gas-tube furnace, and was inclined by means of an iron plate, the gate end being the lower. Details of the construction of the mold are given in Fig. 1.

---

[8] *Trans.*, xxviii., 157 (1898).

A movable clay cover was placed on the crucible so that it could easily be removed, both to determine the "pitch" and to facilitate the addition of the bismuth. The desired amount of bismuth was carefully weighed, placed in a capsule, dropped into the molten copper, and rapidly stirred with a clean red-hot iron rod. As soon as the copper in the crucible was melted the lid was removed from the melting-pot, and a portion of the metal in the crucible was dipped out with a small ladle, previously heated to a red heat so as to prevent chilling and sticking. This sample was allowed to cool and set. If the surface rose, the metal contained too much carbon monoxide; if it

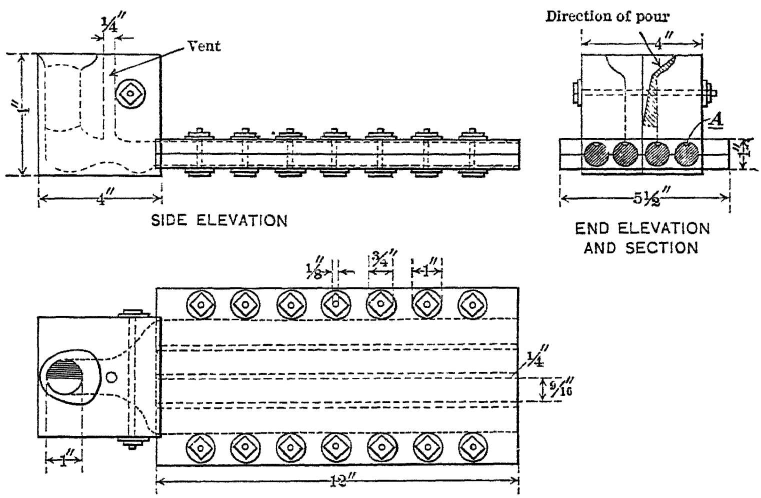


FIG. 1.—DETAIL DRAWING OF MOLD AND GATE.

sunk at some point, it had too much copper oxide; but if the surface neither rose nor sunk the melt was at tough pitch, and on fracturing the sample it would present the typical tough-pitch appearance, being fine-grained to fibrous, free from blow-holes, and of a rose color.

If the first sample showed a gassy melt, the cover of the crucible was removed for a minute or two, and another sample then taken. If the surface of this second sample sunk, the lid was replaced for half the time that it had been previously removed. The next sample should present tough-pitch characteristics. The copper in the casting having the same composition as the molten metal in the crucible just before pouring, it

is absolutely necessary for successful work to determine the pitch accurately. If the copper is "off pitch" at the time of pouring, it is so difficult to re-treat the annealed material that it is practically useless to attempt a remelt.

The testing of the sample by the ladle, and the stirring-in of the cold metallic bismuth, cools the molten copper, and this chilling is augmented by the exposure to the air. It is necessary to have absolute fluidity of the copper for a successful pouring, since the metal has to find its way through the gate and into the fingers of the mold, both of which, being of a lower temperature, tend to freeze the metal before the pour is complete.

Several melts failed on account of not preheating the mold sufficiently. A bright red heat is the proper temperature, which, however, oxidizes the graphite so rapidly that at the end of 10 or 12 melts the mold is worn out.

Rapidity of movement throughout the melt and steadiness of pouring are necessary, especially in pouring the metal into the gate. Here a slight quiver of the hand will cause the copper in the gate or fingers of the mold to separate into patches, the first patch freezing before another gets to it, and forming a "cold-shut" or plane of weakness in the casting. Then, too, one disturbance begets another, and a series of such cold-shuts is the result. Slowness of pouring produces a similar effect, with the added possibility of the mass freezing in the narrow neck of the gate.

With the mold in a vertical position, the air ahead of the pour had a tendency to rise through the bars and become entangled there after the copper had frozen, forming pipes sometimes a third of the length of the casting, or being distributed as blow-holes throughout the entire mass. On approaching the level, instead of being evenly distributed, the blow-holes were found more in the top of the casting, as shown at *A* in Fig. 1. An inclination of about 10° in the opposite direction removed the difficulty arising from the air contained in the mold, since it was pushed out ahead of the molten copper.

A wedge-shaped piece of graphite placed at the top of the pour-hole, like a whistle, tended to prevent any excess air from being entrained at this point, and any air so entrained was allowed to escape through a vent placed at the highest point in

the distributing passage of the gate, Fig. 1. The copper rising against the inclination of the mold gives a better chance for the air to pass up through the vent than if the mold were inclined in the opposite direction, in which latter position the molten mass would be accelerated by gravity.

Despite the care with which the specimens were cast, minute blow-holes occasionally appeared in the fractured section. In such cases an approximation of the area so occupied was made, and the stress recalculated, the final results being comparatively accurate.

Marking the Bars.

The castings were marked as follows: Each melt was given a serial number; subscript numbers 1, 2, 3, and 4 were placed at the lower right side of the serial number to indicate the individual casting of the melt. The serial number is larger in size than the subscript.

Rolling the Bars.

The pieces left of the cast bars, after having been submitted to the tensile test, were heated in a gas-muffle, passed through rolls operated by a small motor, and marked with the original numbers for the purpose of making additional tensile tests. Pure copper castings were first tried, both while hot and cold, the former giving better results. Considering Hampe's remarks on the malleability of copper-bismuth alloys, I presumed that the castings would run through better cold than hot, but experiment proved otherwise. The rolled specimens were freer from checks on the edges if heated before each passage through the rolls. One annealing for the entire reduction in cross-section from the original casting to the final product of the rolls was not as good as one following each of the four steps in the reduction.

Testing the Bars.

Tensile tests were made on both cast and rolled bars, using the Riehlé testing-machine in the mechanical laboratory of Columbia University. The load was applied constantly, using the slow-speed gear; therefore no permanent set was recorded. The balance-beam fell before reaching the point of rupture, which would indicate that the specimen had stretched considerably immediately before it broke. The results of the tests, given in Tables I. and II., were obtained as follows:

If P be the breaking-force in pounds, A the area of cross-section in square inches, then p, the stress in pounds per square inch, will equal P/A. Now, if A be the original area of cross-section and a the area occupied by blow-holes, then p will equal $P/(A-a)$.

TABLE I.—*Physical Properties of Copper and Copper-Bismuth Cast Bars.*

No.	Bi.	Diameter.	Area.	Stress.	Stress.	Elongation.	Area Holes.	Corrected Area.	Corrected Stress.	Remarks.
	Per Cent.	In.	Sq. In.	Lb.	Lb. per Sq. In.	Per Cent.	Per Cent.	Sq. In.	Lb. per Sq. In.	
1_1	0	0.504	0.1995	2,200	11,020	12.5	45.0	0.1097	20,050	Turned down.
1_2	0	0.498	0.1948	2,010	10,320	9.4	55.0	0.0877	22,900	
3_1	0	0.468	0.1720	1,820	10,580	6.2	40.0	0.1032	17,600	
3_2	0	0.430	0.1452	1,910	13,150	7.1	45.0	0.0799	23,900	
6_1	0	0.546	0.2341	3,120	13,320	10.7	35.0	0.1522	20,500	Slightly columnar fracture.
6_2	0	0.548	0.2359	4,540	19,240	12.5	15.0	0.2005	22,600	
6_3	0	0.549	0.2367	4,050	17,110	20.9	20.0	0.1894	21,400	
7_1	0	0.555	0.2419	3,880	16,000	14.6	20.0	0.1935	20,000	Tough ~~pitch~~.
7_2	0	0.552	0.2393	4,540	18,980	16.7	10.0	0.2154	21,100	
8_1	0	0.555	0.2419	4,490	18,560	12.5	15.0	0.2056	21,750	Not as tough as 7_1, 7_2.
8_2	0	0.560	0.2463	4,060	16,480	10.7	20.0	0.1969	20,600	
8_3	0	0.555	0.2419	2,170	8,970	4.7	20.0	0.1935	11,200	
8_4	0	0.560	0.2463	4,590	18,630	10.7	15.0	0.2093	22,000	
9_1	0.2	0.550	0.2376	5,500	23,150	12.5	15.0	0.2020	27,300	Fine-grained fracture.
9_2	0.2	0.550	0.2376	5,680	23,900	12.5	10.0	0.2138	26,500	
9_3	0.2	0.553	0.2402	6,060	25,220	12.5	5.0	0.2282	26,600	
9_4	0.2	0.555	0.2419	5,000	20,660	7.8	8.0	0.2225	22,500	
10_1	0.4	0.555	0.2419	2,500	10,330	5.6	20.0	0.1935	13,000	Columnar fracture.
11_1	0.1	0.553	0.2402	5,180	21,500	15.6	10.0	0.2161	23,900	Fine-grained fracture.
11_2	0.1	0.558	0.2445	1,890	7,730	1.8	50.0	0.1223	15,400	
11_3	0.1	0.560	0.2463	Very	coarse	grain	ed.			Columnar fracture.
12_1	0.05	0.558	0.2445	2,670	9,160	4.2	45.0	0.1345	19,800	
13_1	0.02	0.559	0.2454	3,890	15,850	5.3	35.0	0.1595	24,400	
14_1	0.01	0.563	0.2489	2,210	8,880	6.3	40.0	0.1493	14,300	
14_2	0.01	0.565	0.2507	2,100	8,370	6.3	50.0	0.1253	16,800	
15_1	0.005	0.562	0.2481	2,720	10,960	5.3	25.0	0.1861	14,600	
15_2	0.005	0.570	0.2552	3,100	12,150	7.5	30.0	0.1786	17,300	

TABLE II.—*Physical Properties of Copper and Copper-Bismuth Rolled Bars.*

No.	Dimensions.	Area.	Stress.	Stress.
	Inch.	Sq. In.	Lb.	Lb. per Sq. In.
6_2	0.257 by 0.258	0.0663	2,370	35,700
6_3	0.270 by 0.272	0.0731	2,540	34,600
7_1	0.257 by 0.258	0.0663	2,060	31,100
7_2	0.257 by 0.260	0.0668	2,550	38,300
8_1	0.256 by 0.260	0.0667	2,000	30,000
8_2	0.256 by 0.258	0.0661	1,950	29,500
8_3	0.255 by 0.255	0.0650	2,000	30,750
8_4	0.254 by 0.257	0.0650	2,000	30,750
15_1	0.254 by 0.257	0.0650	2,000	30,750
15_2	0.255 by 0.258	0.0658	1,880	28,600[a]

[a] Check at fracture.

If X be the length in inches of the casting between bearings before being pulled, and Y the length in inches stretched in that distance, then the percentage of elongation will equal Y/X.

The first castings of pure copper were turned down. The fractured section of Nos. 1_1, 1_2, Table I., showed minute blow-holes evenly distributed over the surface. The copper was fine-grained and presented a characteristic color. These specimens were made by the method of the Alloys Research Committee, described in their second report, but being impracticable for my special use it was abandoned. Nos. 3_1 and 3_2 were made by pouring into a vertical mold. On turning down, no imperfections in the way of blow-holes or pipes were met. After fracturing, however, several long holes were exposed, one of which was continuous for a third of the length of the casting. The occurrence of these holes indicated a mechanical disadvantage in having the molds in a vertical position. All subsequent castings were made by the method already described, with the exception of Nos. 6_1 and 6_2, which were poured in the same way but with the mold inclined in the opposite direction. The percentage of blow-holes was reduced, but the fracture was more columnar than granular. The best copper castings produced were Nos. 7_1 and 7_2, which possess a "tough-pitch" fracture. Evidently the condition of pitch at the time of pouring these castings was just right. From the stand-point of pitch, No. 8 was very nearly as good as No. 7. With the exception of the first two melts, in which the area allowed for blow-holes was extremely large, due to the mechanical disadvantage of the method used, the other castings vary in tensile strength according to the pitch, or blow-hole content. For if the pitch be just right there will be no blow-holes present.

Discussion of Results.

Kirkaldy pulled four cast bars of copper of unstated purity, having a diameter of 0.619 in. The stress-average was 24,781 lb. per sq. in., and the elongation was 21.8 per cent. These castings had a granular fracture, which would indicate tough pitch. After allowing for blow-holes, the best results were reduced to about 22,300 lb. per sq. in. These bars were probably cast by dipping from a large mass of molten copper, in which the pitch could be more accurately determined.

No. 9, containing 0.20 per cent. of bismuth, had a tensile strength of 3,500 lb. per sq. in. more than the best copper bars. This melt differs from all other copper-bismuth alloys in that the alloy had a very fine-grained, tough-pitch fracture. Every other casting containing bismuth presented an appearance of long radiating fibers resembling pectolite. This fracture is given in Table I. as columnar, because the radiating fibers are grouped together in the form of columns. The tensile strength of castings of this columnar fracture falls much below that of pure copper and still further below that of No. 9, which was fine grained. Hampe alludes to the difference of strength of the two fractures as follows:

"All alloys of copper with metallic bismuth break easily, and show a coarse-grained, bright fracture, but if they are exceptionally fine grained they offer a much greater resistance to rupture than in the coarse-grained condition."

This difference in fracture is due probably to a difference of pitch at the time of pouring, and it is possible that bismuth itself changes the pitch.

The pieces of castings which were treated by rolling had already been strained by the tensile tests. This strain imparted the tendency to check badly on the edges in passing through the rolls. The annealing-and-rolling process did, however, eliminate the inaccuracy due to the presence of blow-holes in the cast bars. With one exception, all the castings of copper-bismuth either crumbled, split up the middle along a diametric plane, as did No. 9, or were so badly checked on the edges as to render them useless for the tensile tests. The exception was Nos. 15_1 and 15_2, containing 0.005 per cent. of bismuth, which fell 3,000 lb. per sq. in. below the average for pure copper. The fracture of 15_2 showed a check, which was accountable for most of the weakness. Accepting the most accurate result obtained on specimen No. 15_1, there would be but 1,800 lb. per sq. in. drop from the average for pure copper.

Conclusions.

If this difference of fracture, and hence the difference of tensile strength, of copper-bismuth alloys of the same bismuth-content is due to a difference of pitch alone, then we may consider the influence of bismuth on copper when the alloy is

fine grained. For the pitch can be so well controlled in the reverberatory furnace that a tough-pitch melt can always be obtained. Reasoning on this as a basis, an alloy of copper containing 0.18 per cent. of bismuth is stronger than pure copper, the bismuth here presenting the same effect as do arsenic and antimony, its associates in the Periodic Law.

For copper to be rolled, the allowable percentage of bismuth is governed by the quantity which can be present without appreciably lowering the malleability and ductility of the alloy. This limit was found to be less than 0.005 per cent. of bismuth for metal to be rolled, either when hot or cold. If this limit be exceeded, the ductility of the copper is so lowered as to interfere with the process of wire-drawing. It is reasonable to assume that the presence of so small an amount of bismuth will not appreciably lessen the electric conductivity of the copper.

Hampe's limits of the percentage of bismuth which will cause hot- and cold-shortness of the alloy seem to be just the reverse. So that 0.02 per cent. of bismuth makes copper cold-short and 0.05 per cent. hot-short.

Acknowledgment.

My sincere thanks are due to Dr. Myrick N. Bolles, of the Department of Metallurgy, Columbia University, for his guidance and valuable assistance in the work of this thesis; also to Mr. Thompson, of the same department, for making the quantitative determination for bismuth, which served as a check on the synthetic analysis of the alloy. For assistance in conducting the tests on the Riehlé machine, I am indebted to Prof. I. H. Woolson and his assistants in the mechanical laboratory of Columbia University.

The Limit of Fuel-Economy in the Iron Blast-Furnace.

BY N. M. LANGDON, MANCELONA, MICH.

(Spokane Meeting, September, 1909.)

INTRODUCTION.

IN considering the magnificent success of Mr. Gayley's bold experiment of applying dry blast to the blast-furnace, whereby a saving of 20 per cent. of fuel per ton of iron is effected, the question arises whether still further economy in fuel is possible, and, if so, how it is to be attained.

The manner in which the heat generated in the furnace is utilized in the production of pig-iron may be determined, by the method of Bell and Gruner, for any furnace for which the necessary data are obtainable; and the study of the operation, under various conditions, of a number of furnaces, for which such information has been worked out, should enable us to form some conclusion on the subject.

The tables and calculations here given cover the operations of several furnaces reported in our *Transactions*; also the two furnaces using natural and dry blast in Mr. Gayley's experiment, for which the heat-equation has been worked out by Prof. Joseph W. Richards in his admirable discussion[1] of Mr. Gayley's paper, The Application of Dry Air Blast to the Manufacture of Iron, and, finally, various hypothetical furnaces. The full list is as follows:

A. Isabella furnace, with natural blast.

B. Isabella furnace, with dry blast. The data stated for A and B in Table III. are those given by Professor Richards, except that the heat-units are given for one unit of iron instead of 100 units, and some items of heat supplied, as stated by Richards, have been subdivided, the total being the same as given by Richards. (Columns A and B are introduced in Table III. for comparison with C and D.)

---

[1] *Trans.*, xxxvi., 745 to 759 (1906).

C. Isabella furnace, with natural blast, for which I have used my own factors in figuring the heat-units.

D. Isabella furnace, with dry blast, for which my own factors have been used for figuring the heat-units.

E. Hypothetical furnace, in which the conditions are the same as in C, except that the temperature of blast is assumed as high enough to reduce the fuel to the same amount as in D.

E1. Hypothetical furnace. Conditions the same as in D, except that temperature of blast is raised to 1,200° F.

E2. Hypothetical furnace. Conditions the same as in C, except that temperature of blast is 1,200° F. Temperature of gas, radiation, and efficiency of reduction the same as in the dry-blast furnace.

F. Hypothetical furnace. Conditions same as in D, except that temperature of blast is assumed at 75° F.

F1. Hypothetical furnace. Conditions same as in C, except that temperature of blast is assumed at 75° F.

G. Hypothetical furnace. Conditions same as in D, except that the moisture of ore is assumed to have been expelled before charging into the furnace.

G1. Hypothetical furnace. Conditions same as in D, except that the CO_2 of limestone is assumed to have been eliminated before charging.

H. Hypothetical furnace, in which the following conditions are assumed: blast dry; no moisture in the ore; no CO_2 in the limestone; temperature of blast, 1,200° F.; temperature of gas, 376° F.; efficiency of reduction, 86.1 per cent. as in C; radiation, 10 per cent. of total heat developed.

H1. Theoretical furnace, with conditions same as in H, except that the moisture of blast is assumed as in C.

I. Theoretical furnace. Conditions same as in H, except that it is assumed that the nitrogen of the blast has been eliminated.

J. Clarence furnace of Bell Bros.[2]

K. Illinois Steel Co.'s furnace, Union No. 1.[3]

L. Sharon furnace No. 2, in which I conducted an experiment covering a period of about two weeks, as I recollect it, in December, 1898.

M1. Alice furnace, Sharpsville, Pa., from data furnished by

---

[2] *Trans.*, xix., 959 (1890–91). [3] *Trans.*, xx., 287 (1891).

C. I. Rader, manager, in which the efficiency of reduction is assumed at 100 per cent.

M2. Alice furnace, same conditions as M1, except that the efficiency of reduction is assumed at 86.4 per cent., practically the same as in C.

N1. Antrim Iron Co.'s furnace (charcoal), for the month of January, 1901, in which the efficiency of reduction is assumed at 100 per cent.

TABLE I.—*Data Given, Calculated, or (under M1, M2, N1, and N2,) Assumed, for the Furnaces Named.*

		C.	D.	J.	K.	L.	M1.	M2.	N1.	N2.
1.	Capacity, cu. ft.	18,000	18,000	25,500	6,700	....	7,445	7,445	3,416	3,416
2.	Capacity above tuyeres, cu. ft.	16,896	16,896	....	6,424	....	6,666	6,666	3,167	3,167
3.	Hearth area, sq. ft.	143.1	143.1	....	56.7	....	86.6	86.6	33.2	33.2
4.	Output per 24 hr., long tons	358	447	78.6	130	194	236.4	236.4	105.6	105.6
5.	Ore used per ton of iron, long tons.	1.776	1.776	2.40	1.57	1.70	[a]1.662	[a]1.662	2.005	2.005
6.	Limestone, per ton iron, long tons.	0.444	0.444	0.55	0.27	0.589	0.389	0.389	0.151	0.151
7.	Fuel, per ton of iron, long tons	0.958	0.770	1.0	0.779	1.008	0.758	0.758	0.776	0.776
8.	Fuel, per ton of iron, lb.	2,147	1,726	2,240	1,745	2,258	1,698	1,698	1,738	1,738
9.	Temperature of blast, deg. C.	382	465	704	650	560	455	455	427	427
10.	Temperature of gas, deg. C.	281	191	250	125	200	200	200	204	204
11.	Composition of gas by weight.[b] — per cent. CO	1.260	0.8925	1.434	0.943	1.2978	[c]0.7922	[c]0.8801	[c]0.7583	[c]0.8944
12.	Composition of gas by weight.[b] — per cent. CO_2	1.155	1.1275	1.095	0.9845	0.9746	1.1725	[c]1.0344	1.192	[c]0.9779
13.	Moisture per cu. ft. blast, grains	5.66	1.75	....	....	....	....	....	....	....
14.	Moisture per ton of iron, tons	0.045	0.01	0.0315	0.0241	0.0368	0.0366	0.0344	0.025	0.023
15.	Composition of pig-iron. — per cent. iron	95.0	95.0	....	....	94.4	95.5	95.5	95.5	95.5
16.	Composition of pig-iron. — per cent. Si, etc.	1.0	1.0	....	....	2.1	1.0	1.0	1.0	1.0
17.	Composition of pig-iron. — per cent. carbon	4.0	4.0	....	....	3.5	3.5	3.5	3.5	3.5
18.	Composition of ore. — per cent. iron	53.5	53.5	....	....	55.40	55.4	55.4	47.60	47.60
19.	Composition of ore. — per cent. moisture.	10.0	10.0	....	....	8.89	12.0	12.0	10.82	10.82
20.	Composition of ore. — per cent. volatile	0.0	0.0	....	....	2.12	3.0	3.0	2.90	2.90
21.	Composition of ore. — per cent. gangue	13.6	13.6	....	....	....	....	....	....	....
22.	Composition of limestone. — per cent. CO_2	42.6	42.6	....	....	38.84	40.0	40.0	44.0	44.0
23.	Composition of limestone. — per cent. moisture	0.0	0.0	....	....	4.60	....	....	4.0	4.0
24.	Composition of limestone. — per cent. gangue	57.4	57.4	....	....	....	....	....	....	....
25.	Composition of fuel. — per cent. carbon	88.0	88.0	....	....	78.94	86.0	86.0	86.0	86.0
26.	Composition of fuel. — per cent. moisture.	1.0	1.0	....	....	5.40	1.5	1.5	6.0	6.0
27.	Composition of fuel. — per cent. volatile	0.0	0.0	....	....	1.71	1.0	1.0	6.0	6.0
28.	Composition of fuel. — per cent. ash	11.0	11.0	....	....	....	....	....	2.0	2.0

[a] Including 0.102 ton of pig-iron and scrap charged.

[b] In columns A and B (omitted from this table because they are otherwise identical with C and D respectively), the composition of gas is given in percentage by volume as, for A, 22.3 CO, and 13.0 CO_2; for B, 19.9 CO, and 160 CO_2.

[c] Calculated in preparing following tables, and inserted in this table for purposes of comparison.

N2. Antrim Iron Co.'s furnace, with efficiency of reduction assumed at 81 per cent.

Q. Hypothetical furnace. Conditions same as in C, except that the ore is assumed to be protoxide instead of a peroxide.

Q1. Hypothetical furnace. Conditions same as in Q, except that the efficiency of reduction is assumed at 50.8 per cent.

X. Hypothetical furnace, with the following assumptions: charge contains only oxides without slag-making elements;

fuel, pure carbon; temperatures of gas and blast same as in C; iron of the ore is peroxide; blast free from moisture; efficiency of reduction, 100 per cent.

XI. Theoretical furnace, demonstrating the highest theoretical blast-temperature required, with conditions same as in C, so far as they apply, except that efficiency of reduction is assumed at 100 per cent., and temperature of blast as in D.

In presenting data of this kind, it is unfortunate that all writers do not use the same formulas and factors. While the results may be practically the same, comparison is somewhat complicated, and often requires recalculation of the data. In my tables, the data of the Gayley (Isabella) furnaces using natural and dry blast are given first in relative quantities as stated by Richards, and then, together with the similar data worked out for the other furnaces, according to the formulas and factors given by Gruner.

CONSTANTS.

The following are the constants adopted or calculated from Gruner:

Reduction of iron,	weight of O in oxide of iron × 4,400
Reduction of silicon,	weight of O in silicon × 6,125
Fusion of pig,	weight of pig × 310
Fusion of slag,	weight of slag × 500
Expulsion of moisture,	weight of moisture × 606.5
Expulsion of carbonic acid,	weight of CO_2 × 849
Decomposition of H_2O,	weight of H_2O × 3,225
Carried off in gas,	weight of gas × temp. (C.°) × 0.237
Carried in by the blast,	weight of blast × temp. (C.°) × 0.237
Heat generated by combustion:	
C to CO,	weight of C × 2,473
CO to CO_2,	weight of C in CO × 5,607

NOTE.—The weight of O in Fe_2O_3 multiplied by 4,400 is the same as the weight of Fe multiplied by 1,887, as given by Gruner; also the weight of O in silica multiplied by 6,125 is the same as the weight of Si multiplied by 7,000; and the weight of CO_2 in limestone multiplied by 849 is the same as the weight of $CaCO_3$ multiplied by 373.5.

Table I. contains the data, given or assumed, of the actual furnaces on which the succeeding tables are based.

Table II. shows the stock-equation, or materials received into, and discharged from, each of the furnaces named. All of the materials, in the form of ore, limestone, fuel, and blast,

TABLE II.—*Stock-Equation of Materials Received into and Discharged from Blast-Furnaces. In Tons of 2,240 lb.*

Item.	C. Received.		C. Discharged. Iron.	C. Discharged. Slag.	C. Discharged. Gas.	D. Received.	D. Discharged. Iron.	D. Discharged. Slag.	D. Discharged. Gas.
1		Fe	0.95	......	0.407 O		0.95	......	0.407 O
2		Si, etc.	0.01	......	0.011 O		0.01	......	0.011 O
3	Ore......1.776	Gangue	......	0.220	......	Ore......1.776	......	0.220	......
4		Moisture	......	......	0.178 H_2O		......	......	0.178 H_2O
5		Volatile	......	......	......		......	......	......
6		CO_2	......	......	0.190 CO_2		......	......	0.190 CO_2
7	Limestone, 0.444	Gangue	......	0.254	......	Limestone, 0.444	......	0.254	......
8		Moisture	......	......	......		......	......	......
9		Carbon	0.04	......	0.803 C		0.04	......	0.638 C
10	Coke....0.958	Moisture	......	......	0.009 H_2O	Coke....0.770	......	......	0.008 H_2O
11		Volatile	......	......	......		......	......	......
12		Ash	......	0.106	......		......	0.084	......
13	Blast...4.392	Air	......	......	4.347 air	Blast....3.361	......	......	3.351 air
14		Moisture	......	......	0.045 H_2O		......	......	0.010 H_2O
15	Total.......7.570		1.00	0.580	5.990	Total........6.351	1.00	0.558	4.793

Item.	E. Received.		E. Discharged. Iron.	E. Discharged. Slag.	E. Discharged. Gas.	E1. Received.	E1. Discharged. Iron.	E1. Discharged. Slag.	E1. Discharged. Gas.
1		Fe	0.95	......	0.407 O		0.95	......	0.407 O
2		Si, etc.	0.01	......	0.011 O		0.01	......	0.011 O
3	Ore......1.776	Gangue	......	0.220	......	Ore......1.776	......	0.220	......
4		Moisture	......	......	0.178 H_2O		......	......	0.178 H_2O
5		Volatile	......	......	......		......	......	......
6		CO_2	......	......	0.190 CO_2		......	......	0.190 CO_2
7	Limestone, 0.444	Gangue	......	0.254	......	Limestone, 0.444	......	0.254	......
8		Moisture	......	......	......		......	......	......
9		Carbon	0.04	......	0.638 C		0.04	......	0.587 C
10	Coke....0.770	Moisture	......	......	0.008 H_2O	Coke....0.712	......	......	0.007 H_2O
11		Volatile	......	......	......		......	......	......
12		Ash	......	0.084	......		......	0.078	......
13	Blast...3.440	Air	......	......	3.395 air	Blast....3.064	......	......	3.054 air
14		Moisture	......	......	0.045 H_2O		......	......	0.010 H_2O
15	Total. 6.430		1.00	0.558	4.872	Total.......5.996	1.00	0.552	4.444

Item.	F. Received.		F. Discharged. Iron.	F. Discharged. Slag.	F. Discharged. Gas.	G. Received.	G. Discharged. Iron.	G. Discharged. Slag.	G. Discharged. Gas.
1		Fe	0.95	......	0.407 O		0.95	......	0.407 O
2		Si, etc.	0.01	......	0.011 O		0.01	......	0.011 O
3	Ore......1.776	Gangue	......	0.220	......	Ore......1.598	......	0.220	......
4		Moisture	......	......	0.178 H_2O		......	......	......
5		Volatile	......	......	......		......	......	......
6		CO_2	......	......	0.190 CO_2		......	......	0.190 CO_2
7	Limestone, 0.444	Gangue	......	0.254	......	Limestone, 0.444	......	0.254	......
8		Moisture	......	......	......		......	......	......
9		Carbon	0.04	......	0.804 C		0.04	......	0.590 C
10	Coke....0.959	Moisture	......	......	0.009 H_2O	Coke....0.716	......	......	0.007 H_2O
11		Volatile	......	......	......		......	......	......
12		Ash	......	0.106	......		......	0.079	......
13	Blast...4.319	Air	......	......	4.309 air	Blast...3.084	......	......	3.074 air
14		Moisture	......	......	0.010 H_2O		......	......	0.010 H_2O
15	Total.......7.498		1.00	0.580	5.918	Total.......5.842	1.00	0.553	4.289

Item.	H. Received.		H. Discharged. Iron.	H. Discharged. Slag.	H. Discharged. Gas.	I. Received.	I. Discharged. Iron.	I. Discharged. Slag.	I. Discharged. Gas.
1		Fe	0.95	......	0.407 O		0.95	......	0.407 O
2		Si, etc.	0.01	......	0.011 O		0.01	......	0.011 O
3	Ore......1.598	Gangue	......	0.220	......	Ore......1.598	......	0.220	......
4		Moisture	......	......	......		......	......	......
5		Volatile	......	......	......		......	......	......
6		CO_2	......	......	......		......	......	......
7	Limestone, 0.254	Gangue	......	0.254	......	Limestone, 0.254	......	0.254	......
8		Moisture	......	......	......		......	......	......
9		Carbon	0.04	......	0.407 C		0.04	......	0.497 C
10	Coke....0.508	Moisture	......	......	0.005 H_2O	Coke....0.610	......	......	0.006 H_2O
11		Volatile	......	......	......		......	......	......
12		Ash	......	0.056	......		......	0.060	......
13	Blast...2.073	Air	......	......	2.067 air	Blast...0.602	......	......	0.600 air
14		Moisture	......	......	0.006 H_2O		......	......	0.002 H_2O
15	Total.......4.433		1.00	0.530	2.903	Total.......3.064	1.00	0.534	1.523

TABLE II.—*Stock-Equation of Materials Received into and Discharged from Blast-Furnaces. In Tons of 2,240 lb.* (*Continued.*)

J.

Item.	Received.		Discharged. Iron.	Slag.	Gas.
1		Fe............	0.93	......	0.400 O
2		Si, etc.......	0.04	......	0.045 O
3	Ore.........2.40	Gangue.....	......	0.985	....
4		Moisture...	......	......	....
5		Volatile....	......	......	....
6	Limestone,	CO_2..........	......	......	0.242 CO_2
7	0.55	Gangue.....	......	0.308	....
8		Moisture...	......	......	....
9		Carbon.....	0.03	.....	0.847 C
10	Coke.....1.00	Moisture...	......	......	0.025 H_2O
11		Volatile....	......	......	....
12		Ash..........	......	0.098	....
13	Blast.....4.36	Air...........	......	......	4.330 air
14		Moisture...	......	......	0.032 H_2O
15	Total........8.31		1.00	1.391	5.921

K.

Item.	Received.	Discharged. Iron.	Slag.	Gas.
1		0.94	......	0.403 O
2		0.03	......	0.040 O
3	Ore......1.570	......	0.040	....
4		......	......	0.117 H_2O[a]
5		......	......	....
6	Limestone,	......	......	0.119 CO_2
7	0.270	......	0.151	....
8		......	......	....
9		0.03	......	0.640 C
10	Coke....0.779	......	......	....
11		......	......	....
12		......	0.109	
13	Blast...3.132	......	......	3.108 air
14		......	......	0.024 H_2O
15	Total.......5.751	1.00	0.300	4.451

L.

Item.	Received.		Discharged. Iron.	Slag.	Gas.
1		Fe............	0.944	......	0.390 O
2		Si, etc.......	0.021	......	0.019 O
3	Ore......1.700	Gangue.....	......	0.140	....
4		Moisture...	......	......	0.150 H_2O
5		Volatile....	......	......	0.036 vol.
6	Limestone,	CO_2..........	......	......	0.223 CO
7	0.589	Gangue.....	......	0.339	....
8		Moisture...	......	......	0.027 H_2O
9		Carbon.....	0.035	......	0.761 C
10	Coke....1.008	Moisture...	......	......	0.054 H_2O
11		Volatile....	......	......	0.017 vol.
12		Ash..........	......	0.141	....
13	Blast....3.842	Air...........	......	......	3.805 air
14		Moisture...	......	......	0.037 H_2O
15	Total.......7.139		1.000	0.620	5.519

M1.

Item.	Received.	Discharged. Iron.	Slag.	Gas.
1		0.955	......	0.370 O
2		0.010	......	0.011 O
3	Ore......1.560 Iron....0.102	......	0.083	....
4		......	......	0.187 H_2O
5		......	......	0.046 vol.
6	Limestone,	......	......	0.155 CO_2
7	0.389	......	0.234	....
8		......	......	....
9		0.035	......	0.617 C
10	Coke...0.758	......	......	0.011 H_2O
11		......	......	0.007 vol.
12		......	0.088	....
13	Blast...3.552	......	......	3.515 air
14		......	......	0.037 H_2O[a]
15	Total.......6.361	1.000	0.405	4.956

M2.

Item.	Received.		Discharged. Iron.	Slag.	Gas.
1		Fe............	0.955	......	0.370 O
2		Si, etc.......	0.010	......	0.011 O
3	Ore......1.560 Iron....0.102	Gangue.....	......	0.083	....
4		Moisture...	......	......	0.187 H_2O
5		Volatile....	......	......	0.046 vol.
6	Limestone,	CO_2..........	......	......	0.155 CO_2
7	0.389	Gangue.....	......	0.234	....
8		Moisture...	......	......	....
9		Carbon.....	0.035	......	0.617 C
10	Coke....0.758	Moisture...	......	......	0.011 H_2O
11		Volatile....	......	......	0.007 vol.
12		Ash..........	......	0.088	....
13	Blast...3.346	Air...........	......	......	3.311 air
14		Moisture...	......	......	0.035 H_2O
15	Total.......6.155		1.000	0.405	4.75

N1.

Item.	Received.	Discharged. Iron.	Slag.	Gas.
1		0.995	......	0.409 O
2		0.010	......	0.011 O
3	Ore......2.005	......	0.346	....
4		......	......	0.217 H_2O
5		......	......	0.058 vol.
6	Limestone,	......	......	0.066 CO_2
7	0.151	......	0.079	....
8		......	......	0.006 H_2O
9		0.035	......	0.632 C
10	Charcoal,	......	......	0.047 H_2O
11	0.776	......	......	0.047 vol.
12		......	0.015	....
13	Blast...3.648	......	......	3.623 air
14		......	......	0.025 H_2O
15	Total........6.580	1.000	0.440	5.140

N2.

Item.	Received.		Discharged. Iron.	Slag.	Gas.
1		Fe............	0.995	......	0.409 O
2		Si, etc.......	0.010	......	0.010 O
3	Ore.......2.005	Gangue.....	......	0.346	....
4		Moisture...	......	......	0.217 H_2O
5		Volatile....	......	......	0.058 vol.
6	Limestone,	CO_2..........	......	......	0.066 CO_2
7	0.151	Gangue....	......	0.079	....
8		Moisture...	......	......	0.006 H_2O
9		Carbon......	0.035	......	0.632 C
10	Charcoal,	Moisture...	......	......	0.047 H_2O
11	0.776	Volatile....	......	......	0.047 vol.
12		Ash..........	......	0.015	....
13	Blast...3.313	Air...........	......	......	3.286
14		Moisture...	......	......	0.027
15	Total......6.245		1.000	0.440	4.805

[a] Assumed.

TABLE III.—*Heat-Equation, or Heat Required and Supplied.*

Item.		A.	B.	C.	D.	E.	E1.	F.	G.	H.	I.	J.	K.	L.	M1.	M2.	N1.	N2.	X.	Q.	Q1.	G1.	J1.	H1.	E2	E3.	F1.	X1.
	HEAT REQUIRED:																											
1	Reduction of iron.........	1,638	1,638	1,791	1,791	1,791	1,791	1,791	1,791	1,791	1,791	1,760	1,773	1,716	1,628	1,628	1,800	1,800	1,791	1,192	1,192	1,791	1,760	1,791	1,791	1,791	1,791	1,791
2	Reduction of silicon, etc.	68	68	67	67	67	67	67	67	67	67	189	211	94	67	67	53	53	67	67	67	67	189	67	67	67	67	67
3	Expulsion of moisture...	113	113	113	113	113	113	113	4	3	4	15	71	141	120	120	164	161	......	113	114	112	10	3	113	112	115	110
4	Expulsion of carbonic acid..................	180	181	161	161	161	161	161	161	......	......	205	100	189	132	132	56	56	......	161	161	......	101	......	161	161	161	161
5	Expulsion of volatile in fuel.................	......	......	......	......	......	......	......	......	......	......	......	......	38	14	14	94	94	......	......	......	......	......	......	......	......	......	......
6	Expulsion of volatile in ore....................	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......
7	Expulsion of combined water in ore...	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......
8	Fusion of iron...............	250	250	310	310	310	310	310	310	310	310	330	330	330	337	337	300	300	310	310	310	310	330	310	310	310	310	310
9	Fusion of slag...............	232	227	290	279	279	276	290	276	265	271	770	165	341	223	223	220	220	......	286	294	275	165	266	279	273	302	259
10	Decomposition of moisture in blast.....	145	32	145	32	145	32	32	32	20	6	102	78	118	118	111	80	72	......	145	145	32	62	74	145	145	145	58
11	*Total direct*...................	2,626	2,509	2,877	2,753	2,866	2,750	2,764	2,641	2,456	2,449	3,371	2,728	2,967	2,639	2,632	2,767	2,759	2,168	2,274	2,283	2,587	2,617	2,511	2,866	2,859	2,891	2,756
12	Carried off in gas..........	438	238	399	217	229	201	268	202	131	69	350	132	260	235	226	247	231	185	365	405	186	224	138	216	188	490	128
13	Carried off in flue dust.[a]	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......
14	Lost by radiation...........	771	630	582	411	422	402	413	387	288	279	405	525	313	591	374	638	301	419	470	476	378	278	295	420	416	602	320
15	*Total indirect*..............	1,209	868	981	628	651	603	681	589	419	348	755	657	573	826	600	885	532	604	835	881	564	502	433	636	604	1,092	448
16	*Total heat required*.......	3,835	3,377	3,858	3,381	3,517	3,353	3,445	3,230	2,875	2,797	4,126	3,385	3,540	3,465	3,232	3,652	3,291	2,772	3,109	3,164	3,151	3,119	2,944	3,502	3,463	3,983	3,204
	HEAT SUPPLIED:																											
17	Carried in by the blast...	394	399	397	370	464	470	24	339	393	93	727	478	509	383	360	368	334	183	368	409	324	442	413	509	693	31	740
18	From C burned to CO at the tuyeres...........	1,848	1,429	1,881	1,454	1,473	1,326	1,864	1,335	902	1,124	1,928	1,420	1,665	1,526	1,434	1,563	1,419	879	1,734	1,928	1,271	1,206	951	1,436	1,213	2,372	754
19	*Total in zone of fusion*...	2,242	1,808	2,278	1,824	1,937	1,796	1,888	1,674	1,295	1,217	2,655	1,898	2,174	1,909	1,794	1,931	1,753	1,062	2,102	2,337	1,595	1,648	1,364	1,945	1,906	2,403	1,494
20	From C burned to CO....	103	121	105	124	105	124	124	124	105	105	167	163	217	0	93	0	144	......	105	248	123	167	105	124	124	105	......
21	From CO burned to CO_2	1,490	1,448	1,475	1,433	1,475	1,433	1,433	1,433	1,475	1,475	1,304	1,324	1,149	1,556	1,345	1,721	1,394	1,710	902	579	1,433	1,304	1,475	1,433	1,433	1,475	1,710
22	*Total in zone of reduction*	1,593	1,569	1,580	1,557	1,580	1,557	1,557	1,557	1,580	1,580	1,471	1,487	1,366	1,556	1,438	1,721	1,538	1,710	1,007	827	1,556	1,471	1,580	1,557	1,557	1,580	1,710
23	*Total heat supplied*.......	3,835	3,377	3,858	3,381	3,517	3,353	3,445	3,230	2,875	2,797	4,126	3,385	3,540	3,465	3,232	3,652	3,291	2,772	3,109	3,164	3,151	3,119	2,944	3,502	3,463	3,983	3,204

[a] This item is relatively small and has been neglected in these calculations.

which enter into a furnace come out again in the form of pig-iron, slag, flue-dust, and gas, and, when fully accounted for, make an even balance.

Table III. shows the heat-equation for each furnace. All of the heat generated in the furnace by the combustion of the fuel, or taken in with the blast, is fully accounted for and evenly balanced by the heat consumed in the transformation of the materials, or lost by conduction and radiation.

Table IV. contains data calculated for each furnace named.

Remarks on Tables I. to IV.

The blast-furnace may be considered as a heat-engine, the heat from the fuel and blast being consumed in the transformation of the ore, flux, fuel, and blast introduced into the furnace into pig-iron, slag, and gas discharged therefrom, and the inevitable losses inseparable from the operation.

The total heat consumed or required may be subdivided into a number of different items, and the requirement for each item may be traced to its cause, which may be greater or less in one furnace than in another—or in some cases the cause, and consequently the resulting heat-requirement, may be absent; and as the aggregate heat-requirements made up of the different items may be greater or smaller according to the conditions presented, so may the heat-supply from the blast and fuel likewise vary.

The different items which make up the total heat-requirements are given in Table III., items 1 to 16, inclusive. The total of items 1 to 10 as given in item 11 is the direct heat-requirement. That is to say, the heat for each item is in direct proportion to the quantity of the substance for which it is required, contained in the materials introduced into the furnace, as given in Table II.

The total of items 12, 13, and 14, as given in item 15, Table III., is the indirect heat-requirement. It is affected directly by any variation of the total direct heat-requirement, temperature of the blast, gas, etc.; and is inversely proportional to the efficiency with which the fuel is consumed.

Table III. shows that items 1 to 8, inclusive, of heat-requirement are precisely the same for furnace C, using natural blast, and furnace D, using dry blast.

TABLE IV.—*Data of Operation, Efficiency, Etc., Calculated for the Furnaces Named.*

	C.	D.	E.	E1.	F.	G.	H.	I.	J.	K.	L.	M1.	M2.	N1.	N2.	X.	Q.	Q1.	G1.	J1.	H1.	E2.	E3.	F1.	X1.
Carbon of CO burned to CO_2 in reducing-zone, tons	0.263	0.2555	0.263	0.2555	0.2555	0.2555	0.263	0.263	0.2325	0.236	0.2049	0.2775	0.2398	0.307	0.2487	0.305	0.1609	0.1033	0.2555	0.2325	0.2630	0.2555	0.2555	0.2630	0.305
Carbon of CO possible to have burned to CO_2 in reducing-zone, tons	0.3054	0.3054	0.3054	0.3054	0.3054	0.3054	0.3054	0.3054	0.3000	0.302	0.2925	0.2775	0.2775	0.307	0.307	0.305	0.2033	0.2033	0.3053	0.3000	0.3053	0.3053	0.3053	0.3053	0.305
Efficiency of reduction, per cent	86.1	83.7	86.1	83.7	83.7	83.7	86.1	86.1	77.5	78.1	70.0	100.0	86.4	100.0	81.0	100	79.1	50.8	83.7	77.5	87.1	83.7	83.7	86.1	100
Carbon burned to CO in reducing-zone, tons	0.0424	0.0499	0.0424	0.0499	0.0499	0.0499	0.0424	0.0424	0.0675	0.066	0.0876	0	0.0377	0	0.0583	0	0.0424	0.1000	0.0499	0.0675	0.0424	0.0499	0.0499	0.0424	0
Carbonic oxide (CO) in gas, tons	1.26	0.8925	0.875	0.7726	1.246	0.7805	0.336	0.546	1.434	0.943	1.2978	0.7922	0.8801	0.7583	0.8944	0.1181	1.359	1.811	0.7196	0.7525	0.3830	0.8752	0.6648	1.7239	0
Carbonic acid (CO_2) in gas, tons	1.155	1.1275	1.155	1.1275	1.1275	1.1275	0.9643	0.9643	1.0945	0.9845	0.9746	1.1725	1.0344	1.192	0.9779	1.1183	0.7806	0.5694	0.9368	0.9720	0.9643	1.1275	1.1275	1.1550	0.1309
Ratio $\frac{CO_2}{CO}$ by weight	0.916	1.263	1.32	1.4594	0.905	1.444	2.870	1.766	0.763	1.044	0.751	1.480	1.1753	1.572	1.0933	9.4691	0.5735	0.3144	1.3018	1.3180	2.5180	1.2800	1.6960	0.6700	0
Efficiency of utilization of fuel, per cent	93.5	91.5	92.8	92.3	92.4	91.2	91.3	92.0	91.6	90.1	87.8	100.0	93.9	100.0	90.9	100	92.	83.1	91.0	87.6	91.4	92.6	92.5	94.4	100
Active capacity (above tuyeres), cu. ft	16,896	16,896	16,896	16,896	16,896	16,896	16,896	16,896	....	6,424	....	6,666	6,666	3,167	3,167	16,896	16,896	16,896	16,896	....	16,896	16,896	16,896	16,896	16,896
Hearth-area, sq. ft	143.1	143.1	143.1	143.1	143.1	143.1	143.1	143.1	....	56.7	63.6	86.6	86.6	33.2	33.2	143.1	143.1	143.1	143.1	....	143.1	143.1	143.1	143.1	143.1
Ratio of hearth-area to active capacity, 1 to	118	118	118	118	118	118	118	118	....	113	....	77	77	95.4	95.4	118	118	118	118	....	118	118	118	118	118
Output of iron per 24 hr., tons	358	447	447	484	359	481	678	564	78.6	130	194	236.4	236.4	105.6	105.6	768	385	328	502	....	647	450	520	290	875
Fuel consumed per ton of iron, lb	2,147	1,726	1,726	1,595	2,148	1,604	1,138	1,367	2,240	1,745	2,258	1,698	1,698	1,738	1,738	[a]886	1,994	2,341	1,537	1,494	1,189	1,707	1,478	2,652	878
Rate of driving: lb. C burned in 24 hr. per cu. ft	36.0	35	35	34.4	36	34	33	34	5.48	26	....	49.0	46.0	47.2	42.9	36	36	34	34	....	33	35	34	37	35
Rate of driving: lb. C burned in 24 hr. per sq. ft. hearth-area	4,265	4,118	4,170	4,066	4,240	4,066	3,872	4,016	....	2,948	4,601	3,773	3,542	4,500	4,087	4,277	4,226	4,005	4,041	....	3,900	4,093	4,000	4,358	4,181

[a] Carbon, equivalent to 1,006 lb. coke.

Tabulating the differences shown in Table III., in items 9, 10, 12, and 14 of heat-requirements, and 17, 18, 20, and 21 of heat-supply, between furnaces C and D, we have the figures given in Table V.

TABLE V.—*Comparison of Furnaces C and D (Items* 9, 10, 12, 14, 17, 18, 20, *and* 21, *Table III.), Showing Difference in Heat-Units and Its Percentage of the Total Heat* (3,858 *B.t.u.*) *of C.* (See Item 16, Table III.)

Item.		C.	D.	Difference.	Per Cent.
		Heat-Units.			
	HEAT REQUIRED:				
9.	For fusion of slag	290	279	11	0.28
10.	For decomposition of moisture in blast	145	32	113	2.93
	Direct saving in heat-requirements by D	....	....	124	3.21
12.	Carried off in gas	399	217	182	4.72
14.	Carried off by radiation	582	411	171	4.43
	Indirect saving in heat-requirements by D	....	....	353	9.15
	Total saving in heat-requirements by D	....	....	477	12.36
	HEAT SUPPLIED:				
17.	Carried in by blast	397	370	27	0.70
18.	From combustion of C to CO at tuyeres	1,881	1,454	427	11.06
	Difference in zone of fusion	....	....	454	11.76
20.	From combustion of C to CO in reducing-zone	105	124	−19[a]	−0.49[a]
21.	From combustion of CO to CO_2 in reducing-zone	1,475	1,433	42	1.09
	Difference in zone of reduction	....	....	23	0.60
	Total difference in heat-units supplied	....	....	477	12.36

[a] In this one item, the figure for D is larger than for C, and therefore is entered with a minus sign.

It will be noted that the furnaces after K in the foregoing list, and in some of the tables, are not discussed in the text. This is due to the circumstance that the tables were originally prepared to accompany a wider consideration of blast-furnace theory, in which the effect of hypothetical changes in dimensions, etc., was to be considered. This more general discussion has been postponed to a future opportunity, so as to base the

present paper chiefly upon data of actual practice. But, at the suggestion of the Secretary, the apparently superfluous columns have been retained in the tables as valuable for further study and reference.

Remarks on Table V.

The small decrease of 11 units in heat required by D under item 9 is due to the saving in total heat-requirements, which involves less fuel, and consequently the presence of less slag-making material, in the furnace using dry blast, as may be seen by comparing item 11 in columns C and D, Table III.

The saving of 113 units under item 10 is due directly to the decreased amount of moisture in the blast of furnace D, using dry air. It will be noted that this saving by itself is only 2.93 per cent., and added to the small item of 0.28 per cent. saved in the fusion of the slag, makes a total direct saving in heat-requirements of only 124 heat-units, or 3.21 per cent.

The saving in heat carried off in the gas (item 12) is 182 heat-units, or 4.72 per cent. of the total heat supplied. This saving is due indirectly to the decreased moisture in the dry-blast furnace D, and directly to the decreased quantity of blast required for the combustion of the decreased quantity of fuel, and the consequent decreased weight of gas discharged; also to the lower temperature of the escaping gas from the dry-blast furnace, which is a direct result of the decreased weight of gas.

In item 14 (radiation), 171 heat-units, or 4.43 per cent. of the total heat-requirements of the furnace using natural blast, are saved by D. It is reasonable to assume that the amount of heat radiated from the furnace is proportional to the temperature and the quantity of heat developed, and that it would vary at different parts of the furnace.

We have in Table III. (item 23) the total heat supplied to furnaces C and D, as well as the portion developed in the fusion-zone (item 19) and the reduction-zone (item 22). The temperature of the fusion-zone, as measured by the composition of the iron and slag, was the same in both furnaces. In the reducing-zone the temperature, as measured by that of the gas discharged, was considerably lower in D (dry blast); therefore this furnace should radiate proportionally less heat than C (natural blast). This is exhibited in Table III., which shows the total loss by radiation in C to be 582 units, or 15.1 per

cent. of the total heat (3,858 units) supplied, while the loss in D was only 411 units, or 12.2 per cent. of the total heat (3,381 units) supplied. As compared with the fusion-zone the loss by radiation in C was 25.5 per cent. of the total (2,278 units) supplied to that zone, while in D it was 22.5 per cent. of the total heat (1,824 units) supplied to that zone. Finally, as compared with the reduction-zone, C lost by radiation 36.8 per cent. of the 1,580 units supplied to that zone, while D lost 26.4 per cent. of the 1,557 units supplied. The comparison thus shows a saving in heat lost by radiation by D of 2.9 per cent. of the total heat supplied, 3 per cent. of the heat of the fusion-zone, and 10.4 per cent. of the heat of the reduction-zone.

The difference of 3 per cent. between C and D in the percentage lost by radiation relative to the heat supplied to the fusion-zone is somewhat above the truth, for the reason that a part of the radiation from D (dry blast) was affected by the decreased temperature of the reducing-zone. It is, however, reasonable to assume that the actual error, if any, is well within the limits of permissible error in calculations of this nature.

The difference of 10.4 per cent. relative to the heat developed in the reducing-zone shows plainly the effect of the cooler top, as well as that of the decreased quantity of heat developed in this zone of the dry-blast furnace.

It is interesting to note in Table V. that the total indirect saving of heat-requirements in the dry-blast furnace is 353 heat-units, or 9.15 per cent.—nearly three times as great as the direct saving; and that, while the total saving in heat-requirements is only 477 heat-units, or 12.36 per cent., yet (according to Table IV.) the actual saving in fuel is 421 lb. per ton, or 19.61 per cent. of the 2,147 lb. per ton of pig-iron used in C with natural blast.

Right here is the *crux* of the question, how and why the direct saving of 2.93 per cent. in heat-requirements for the decomposition of a small quantity of moisture in the blast should effect such a startling saving in the fuel required per ton of iron produced.

Of the fact that there is a saving of about 20 per cent. in fuel there can be no doubt; and many theories have been advanced to account for it.

THEORIES OF THE ECONOMY OF THE DRY BLAST.

Three theories have been advanced, either of which apparently accounts for the saving: "Additional uniformity obtained by the use of dry air," first advanced by Gayley[4] and later emphasized by Raymond[5] and others; "critical temperature," advanced by Johnson[6] and supported by Howe;[7] "more oxygen per day and increased reducing power and smelting-down capacity," advanced by Richards.[8] To these I will add another and very simple reason—namely, that there is less work to be done in the furnace using dry air, and that, as a natural consequence, less fuel is required to do it.

Have those who advance the "uniformity" theory mistaken the shadow for the substance? Applied to the operation of the blast-furnace, uniformity is descriptive of a condition and is not itself a condition. Applied to the moisture of the blast, it is descriptive of its quantity. Taken in this connection it also means uniformly low quantity of moisture. In accounting for the saving of fuel with dry, as compared to natural blast, by the more uniform condition of the moisture of the blast, it is apparent that the full meaning of the term has been lost sight of. There is not the least doubt that uniformity in the moisture of the blast contributes largely to uniformity in the working of the furnace and the condition of its product; but since, in the case of both the natural- and the dry-blast furnaces the working as well as the product is considered at an average, "uniformity" has no significance in accounting for the saving in fuel. Can any one familiar with the phenomena of blast-furnace operation doubt but that, with equal uniformity of the moisture in the blast at its maximum instead of its minimum quantity, there would have been equal uniformity in the working and product of the furnace using moist blast accompanied with higher instead of lower fuel?

Concerning the "critical temperature" theory, there may or may not be such a condition as applied to the working of the blast-furnace. It is evident, however, that there must be a sufficient temperature. From the data of the working and

[4] *Trans.*, xxxv., 771 (1905).
[5] *Trans.*, xxxv., 1023 (1905).
[6] *Trans.*, xxxvi., 472 (1906).
[7] *Trans.*, xxxvii., 216 (1907).
[8] *Trans.*, xxxvi., 745 *et seq.* (1906); xxxvii., 224 (1907).

product of the two furnaces reported by Gayley, we know that the temperature, critical or otherwise, was sufficient in each case. Although we do not know the degree of temperature, we do know, what is of the utmost importance, that it was precisely the same and is accurately measured by the composition of the iron and the slag from each furnace—therefore, this theory will not account for the results obtained by the use of dry blast.

As to more oxygen per day and faster driving, Richards says:[9]

"Weight of oxygen present as air (natural blast), 96.3 kg. per 100 kg. of iron," equivalent to 0.963 ton per ton of iron, which, multiplied by the daily output of 358 tons, gives oxygen per day, 344.75 tons.

"Weight of oxygen present as air (dry blast), 76.5 kg. per 100 kg. of iron," equivalent to 0.765 ton per ton of iron, which, for daily output of 447 tons, gives oxygen per day, 341.95 tons; difference, 2.80 tons.

Or only 0.71 per cent. Evidently this small difference cannot account for the difference in results.

"Increased reducing power" is given as a reason. This is a mistake also. In Table IV. I have given the following conclusions, computed from data given by Richards:

Carbon of CO, Burned to CO_2 in Reducing-Zone, Per Ton of Iron.

	Ton.
Natural blast (Furnace C),	0.263
Dry blast (Furnace D),	0.2555
Difference in favor of natural blast,	0.0075

"Increased smelting-capacity" is the next reason advanced. Right here is the nut; and I think Richards has mistaken the husk for the kernel. The converse of his statement is the true reason; that is to say, less smelting-capacity is required.

In Table V. it is seen how the direct saving in heat-requirement for the decomposition of the moisture of the blast of 113 heat-units, or 2.93 per cent., is increased a little by the slag-smelting requirement, and I have explained how this is still further increased nearly three-fold by the saving in heat carried off by the gases and by radiation, until we have a total saving of 477 heat-units, or 12.36 per cent. It remains to be shown how

[9] *Trans.*, xxxvi., 749 (1906).

this saving of 12.36 per cent. of heat-requirements can effect a saving of 19.6 per cent. in fuel. Item 17, Table V., shows a saving of heat supplied at the tuyeres of 27 units. This is the heat carried in by the blast and does not affect the fuel directly. Item 18, a saving of 427 units, is from the combustion of the fuel to CO and is equivalent in coke to: $\frac{427}{2473} \times \frac{2240}{0.88} = 440$ lb. per ton of iron. In the reducing-zone there were 42 units less developed in the dry-blast furnace from the combustion of CO to CO_2, and 19 units of this must be made up by the combustion of solid fuel to CO, which is equivalent in coke to $\frac{19}{2473} \times \frac{2240}{0.88} = 19$ lb. per ton of iron. The difference, which is saving in coke effected by the use of the dry blast, is $440 - 19 = 421$ lb. per ton of iron.

It will be noted that the saving in direct heat-requirements by the use of dry blast is wholly in the smelting-zone, and that, if reduction had been as effective in the dry- as in the natural-blast furnace, the saving would have been 440 lb. of coke.

Notwithstanding the increased temperature of the dry blast, there was, owing to the decreased quantity of blast required per ton of iron, an actual decrease of 27 heat-units carried into the dry-blast furnace, which had to be supplied with coke burned to CO equal to $\frac{27}{2473} \times \frac{2240}{0.88} = 28$ lb. per ton of iron, so that, if all the conditions except that of moisture had been precisely the same in the dry-blast as in the natural-blast furnace, a total saving of 468 lb. of coke per ton of iron might have been effected.

Several writers who have found it difficult to account for the saving of such a large amount of coke as compared with the quantity theoretically required for the dissociation of the moisture eliminated, have evidently assumed, in their theoretical calculations, the complete combustion of the coke to CO_2, overlooking the fact that, owing to the nature of the blast-furnace process, the moisture of the blast would be dissociated at the expense of coke burned to CO, or incomplete combustion, requiring more than 3.25 times as much coke as they deem to be required.

Comparative Results.

The evolution of the blast-furnace, from the primitive form and practice, which consumed more than 7 tons of fuel per ton of iron, to the present form and practice, in which the fuel has been reduced to less than 1,800 lb. per ton of iron, has been marked by five distinct epochs, each of which has produced startling immediate results, as well as permanent changes. The coking of bituminous coal before charging into the furnace, Neilson's application of the hot blast, and Bell's discovery of the relation of the form of the furnace to the proper reduction of the ores, marked three of these epochs. The next profound change, somewhat less startling in its approach and effect, had its maximum development in the wide distribution of the exceptionally rich ores of the Lake Superior region, followed almost immediately by the largely increased supply of artificially-concentrated magnetites. The last epoch of progress, but not the least important in revolutionary results, has been inaugurated by Gayley's process of refrigerating the blast for the purpose of eliminating its moisture.

It is interesting to note that, of the causes above named as contributing to the reduction of the amount of fuel required to make a ton of iron, only one—that of Bell—involves any change in our view of the nature of the blast-furnace process, or of the reactions taking place within the furnace; all the others pertaining simply to a better preparation of the materials (ore, blast, fuel, etc.) before they enter the furnace. The preliminary coking of the coal eliminated the volatile hydrocarbons and concentrated the fixed carbon, which only is of value in the furnace. Heating the blast by means of the escaping gas, which has performed its function in the furnace, recovers a portion of its potential heat and returns it to the furnace at the point where, owing to the nature of the process, it is most effective in oxidizing the solid fuel. Concentration of the ores, whether natural or artificial, as well as the elimination of the moisture from the blast, reduces the heat-requirements and therefore the fuel-consumption.

According to the principles already known, it may be asserted, without any claim to the gift of prophecy, that there are to-day apparently no reasons, except commercial ones, why

iron should not be produced in the blast-furnace with 1,200 lb. or less of coke per ton. In the tables given with this paper, furnace E is a hypothetical one, having the same conditions as natural-blast furnace C, except that the temperature of the blast is increased to a point where it will be equivalent in results to those effected in D (dry blast). The temperature found to be sufficient is 569° C. (1,056° F.). It will be noted, moreover, that in Table III. the items of heat-requirements from 1 to 9, inclusive, are precisely the same in E as in the dry-blast furnace, item 9 being smaller for E than for C (natural blast), for the same reason as made it smaller with D—namely, less fuel, and consequently somewhat less slag. Item 10 (the decomposition of the moisture) would, of course, be the same for E as for C (natural blast). It is in items 12 and 14 that the significant changes in heat-requirements occur; the heat carried off in the gas being, for E, lower than for C (natural blast), and somewhat higher than for D (dry blast). The heat-conditions for radiation are similarly affected.

As compared with C (natural blast), it will be noted that there is a small direct saving in heat-requirements (item 9) of 11 units, or 0.2 per cent., and an indirect saving (items 12 and 14) of 330 units, or 8.6 per cent., to which is to be added the increased amount of heat carried in by the blast (item 17), 67 units, or 1.7 per cent., making a total saving of 408 units, or 10.5 per cent., in the heat to be developed from the fuel as per item 18, which gives 1,881 units for C and 1,473 for E; difference, 408 units, or 10.5 per cent.

As the heat in item 18 comes from carbon burned to CO in the fusion-zone, the fuel saved would be $\frac{408}{2473} \times \frac{2240}{0.88} =$ 241 pounds.

Had the stove-equipment for C been of ample capacity, the application of the dry-air blast would have shown a still more remarkable decrease in fuel-consumption. This is demonstrated in hypothetical furnace E1, in which the conditions are the same as in D (dry blast), except that the temperature of the blast is assumed to be 1,200° F. (650° C.), a temperature easily and safely maintainable with fire-brick stoves.

It will be noted in Table III., item 11, that there is a difference of only 3 units in the total of direct heat-requirements

between D (dry blast) and E1, representing the same conditions, except that the temperature of the blast is 330° F. (185° C.) higher. This small difference is due to the decrease in fuel, and consequently in the amount of slag from the ash, as shown in Table II.

In the indirect heat-requirements a saving for E1 of 16 units over D, carried off in the gas (item 12, Table III.), is due to the smaller quantity of gas; and that of 9 units in radiation (item 14) is due to the smaller amount of heat developed and, as a natural consequence, less radiation. This makes a total saving for E1 in heat-units required of 28. Adding the increased heat carried in by the blast (470 — 370 = 100 units), we have a total of smaller heat-requirement from the fuel of 128 units. This agrees with item 18 (1,454 — 1,326 = 128 units), which is heat developed by the combustion of C to CO; therefore, the equivalent in coke is $\frac{128}{2473} \times \frac{2240}{0.88} = 131$ lb. saved by E1 over D, by reason o the increased temperature of the blast. The corresponding saving of E1 over the natural-blast furnace C would be $421 + 131 = 552$ lb. of coke.

Furnace F shows what might be expected with atmospheric temperature 75° F. (24° C.), other conditions being the same as with dry blast. As might be expected, the heat-requirement for fusion of slag (Table III., item 9) is, by reason of increased ash from the increased quantity of fuel required, greater than in either D, E, or E1, and exactly equals the requirement in C (natural blast), as it should do, because, as will be seen later, the fuel required is precisely the same.

Under indirect heat-requirements, item 12 (carried off in the gas) is greater than in D, E, or E1, but considerably less than in C. The temperature of the gas being the same as in D or E1, while the quantity is considerably greater, the heat-requirement is, of course, greater. As compared with C and E, the temperature of the gas from F is lower, and the increased quantity of gas more than balances the effect of this factor in the case of E, but is not sufficient to do so in the case of C.

It will be noticed that the requirement for radiation (Table III., item 14) varies directly and almost in exact ratio with the heat developed in the fusion-zone of C, D, E, and E1. This was

to be expected, since the heat developed in the reducing-zone is nearly the same in all these furnaces.

Furnace G shows what might have been expected if, in addition to drying the blast, the moisture had been expelled from the ore before charging into the furnace. The saving in heat-requirements is shown as follows in Table III.:

	Heat-Units.
Direct saving:	
Item 3, expulsion of moisture,	109
Item 9, fusion of slag,	3
Indirect saving:	
Item 12, carried off in gas,	15
Item 14, loss by radiation,	24
Total saving in heat-requirements,	151

From this total saving must be deducted the decreased heat carried in by the blast (item 17), 31 units, leaving a total net saving in heat-requirements of 120 units.

Since the saving in heat-requirements all comes from fuel burned to CO, the equivalent in coke would be $\frac{120}{2473} \times \frac{2240}{0.88} =$ 122 lb. of coke less than would be required per ton of iron by D.

The conditions of furnace H are: dry blast; moisture of ore and carbonic acid of limestone expelled by calcination before charging; temperature of blast, 1,200° F. (650° C.); temperature of gas, 375° F. (191° C.); radiation, 10 per cent. of total heat developed. All of these conditions are practicable at the present time to the degree stated, and some of them could be made still more favorable.

Comparing furnace H with C (natural blast), it will be seen, Table III., that under direct heat-requirements, those of the reduction of iron (item 1), and silicon (item 2), and the fusion of iron (item 8), are precisely the same. Since, under H, the moisture in the ore has been expelled, the only heat-requirement of this nature is 3 units (item 3) for the moisture contained in the coke. Item 4, for carbonic acid, drops out. Less fuel, and consequently less ash and slag, reduces item 9; and, of course, as we have seen, the dry blast makes less requirement for decomposition of the moisture in the blast (item 10). Altogether there is a total saving in direct heat-requirements of 421 units, or 10.91 per cent. of the total for C.

In indirect requirements there is a saving in heat carried off in gas (item 12) of 268 units, or 6.95 per cent. This is due to the large decrease in the quantity of gas (Table II., item 15), amounting to $\frac{5.990 - 2.903}{5.990} = 51.5$ per cent., which, in turn, is due to the decreased quantity of fuel and blast required, and to the lower temperature of the gas. In the item of radiation (Table III., item 14) there is a decrease of 294 units = 7.62 per cent. This is due to the decreased quantity of total heat supplied (item 23), and also to the lower temperature in the reducing-zone, as evidenced by the lower temperature of the escaping gas.

As to heat supplied in the fusion-zone, that carried in by the blast (item 17) for H is 4 units, or 0.10 per cent., less than in C (natural blast), notwithstanding the large increase in temperature. This is due to the fact that the decrease in quantity of blast overcomes the gain from increased temperature. This item deducted from the total decreased heat-requirements represents the total decrease of heat required to be supplied from the fuel.

Tabulating these results, we have:

Item No.	Units.	Per Cent.
11, saving in direct requirement, ...	421 =	10.91
12, saving in heat carried off by gas, ...	268 =	6.95
14, saving in radiation, ...	294 =	7.62
	983	25.48
17, deduct less heat carried in by the blast, . .	4 =	0.10
Total net saving of heat-requirements, ...	979 =	25.38

This all comes from fuel burned at the tuyeres, and is shown in item 18 (1881 — 902 = 979 units), being equivalent in coke to $\frac{979}{2473} \times \frac{2240}{0.88} = 1{,}009$ lb., saving in coke per ton of pig-iron as compared with natural blast, or a total of 1,138 lb. of coke per ton of iron.

The conditions of furnace I are precisely the same as those of H, except that the nitrogen has been eliminated from the blast. Comparing with H, it is seen (Table III.) that the heat-requirement for the expulsion of moisture (item 3) is increased 1 unit, due to the increased fuel required, and the requirement for the fusion of slag (item 9) is increased 6 units for the same

reason, while the requirement for decomposition of the moisture in the blast (item 10) is decreased 14 units, making a total net decrease in direct heat-requirements (item 11) of 7 units.

Under indirect heat-requirements, item 12 (carried off in gas) is reduced 62 units by reason of the decreased quantity of gas, and item 14 (radiation) is reduced 9 units by reason of the decreased quantity of heat supplied (item 23).

Of the heat supplied, there were 300 units less carried by the blast into I than into H (item 17, Table III.).

Tabulating these results, we have:

Comparison of H with I.

Item No.		Heat-Units.
17, less heat carried by the blast with H, ...		300
11, decrease in direct requirements, ...		7
		293
12, saving in heat carried off in gas, ...	62	
14, saving in radiation, ...	9	71
18, increased heat required (1,124 — 902), ...		222

This heat derived from fuel burned to CO at the tuyeres would be equivalent in coke to $\frac{222}{2473} \times \frac{2240}{0.88} = 229$ lb. more than in furnace H, or, in other words, I would require 1,367 lb. of coke per ton of iron, showing that the elimination of nitrogen from the blast, instead of reducing, would increase the fuel-consumption.

Furnace J, the Clarence furnace of Bell Brothers, shows strikingly how the use of lean ores affects the fuel-consumption. Table I. shows that in this furnace 2.4 tons of ore were required for 1 ton of iron, requiring 0.55 ton of limestone as flux, and producing from these materials, together with the ash from the coke (Table II., item 15 under J), 1.391 tons of slag as against 0.58 ton in the natural-blast furnace C, while requiring in direct heat for the one item of the fusion of slag (Table III., item 9) an increase of 770 — 290 = 480 heat-units, equivalent to an increase in coke of $\frac{480}{2473} \times \frac{2240}{0.88} = 482$ lb. per ton of pig-iron.

Furnace K, Union furnace No. 1 of the Illinois Steel Co., shows the effect of using a rich ore, only 1.57 tons of ore (Table I., K, 5) and 0.27 ton of limestone (Table I., K, 6) being used and, together with the ash from the fuel, producing only 0.3 ton

of slag (Table II., K, 15) per ton of iron, thus saving in direct heat-requirement for the fusion of slag (Table III., item 9) $290 - 165 = 125$ units, equivalent in coke to $\frac{125}{2473} \times \frac{2240}{0.88} =$ 129 pounds.

Conclusions.

Upon careful consideration of the foregoing facts and figures, it will be evident that there are no more startling economies in fuel-consumption by the iron blast-furnace to be achieved by a further improvement in the air-blast, elimination of the nitrogen having been shown to be a detriment instead of a benefit.

With conditions already practically reducing the coke to about 1,200 lb. per ton of iron, the saving to be effected by carrying a blast-temperature above 1,200° F., say at 1,600° F., would be considerable; but, since reserve heat must be carried somewhere for emergencies, it is doubtful whether it would be advisable in practice to carry regularly more than 1,200° F.

Improvements in practice will no doubt enable us to reduce the moisture of the blast somewhat below 1.75 grains per cubic foot, and thus effect some additional saving in fuel. But more important future progress in the economy of blast-furnace fuel must be the result of a more careful preparation of the ore, limestone, and fuel, eliminating from these materials the moisture and volatile matter, and reducing the slag-making elements. Of course, the cost of such preparation will have to be weighed against the resultant saving of fuel in the furnace-process itself, and the scientific furnace-manager will find the problem with which he has to deal not less difficult than were the cruder problems of a more ignorant practice. In fact, as we all know, the result of scientific progress in our art is to make our responsibility greater and our task more complex. Pioneers like Mr. Gayley certainly confer great benefits upon capitalists, workingmen, and the public at large; but it cannot be denied that they worry their professional colleagues!

Need of Instrumental Surveying in Practical Geology.

BY BENJAMIN SMITH LYMAN, PHILADELPHIA, PA.

(Spokane Meeting, September, 1909.)

THERE seems to be dire need of repeated preachment against the too-frequent sad neglect of instrumental surveying and mapping in geological surveys. The value of the map as an illustration of the statements and opinions of a report is too apt to be overlooked; and its essential necessity in working out during its construction the proper conclusions from the observed facts is generally altogether misunderstood.

Practical geology seeks, of course, to ascertain and indicate the character of workable or unworkable beds or veins, their depth, position, dip, and horizontal course, or strike, even below the surface; also their outcrops, and therefore, if workable, their extent; and so, taking account of their thickness and specific gravity, their weight in tons. It might, therefore, be called quantitative geology. For the study of more abstruse geological questions, too, it is in many cases necessary to gain some knowledge, or reasonable opinion, in regard to such hidden facts. In some cases, careful instrumental surveys are made; in others, from necessity or choice, there are few or no instrumental observations.

It was said, 15 or 20 years ago, that there was an engineer in the anthracite-regions who was capable of telling, on first glancing at a new place, exactly what coal-bed was to be found there at a certain depth, and how thick—a splendid second-sight! "'O, there be,'" you may be tempted to exclaim, "geologists 'that I have seen' geologize 'and heard others praise, and that highly, not to speak it profanely,'"—but let them pass without being fully characterized, lest the temptation towards profanity be altogether too strong. Yet such second-sight does not exceed the expectations of many men of imperfect geological knowledge, men who have heard of marvelously successful geological predictions, but are unaware of

the methods by which they were arrived at. The marvel in such cases, however, is not seldom much exaggerated, and is sometimes altogether imaginary. On one occasion my Japanese assistants gleefully told of a joyful oil-well digger who declared that I had dramatically stamped my foot on the ground at a certain spot and told him to dig there for oil; he had done so, and had been very successful. The story at first seemed wholly a mistake, but, under reflection, proved to have the foundation that he had been told that at a point indicated, a few yards from where he was digging at the outcrop of a steeply-dipping oil-bearing bed, he would find the same bed at a certain depth and probably less drained of its oil. There had been, after all, nothing magical about it, nothing whatever smacking of the divining-rod. It is true that, with or without any such implement or paraphernalia of any kind, but perhaps with much undisplayed knowledge of the subject and of kindred circumstances, and with keen, yet instantaneous, "lightning-calculating" observation of the conditions present, shrewd estimates and sagacious guesses are sometimes made; and this may account for the successes of some of the divining-rod men. The public (including, too, men like deep-hole drillers), somewhat acquainted with underground work, surprised, as they are, at the frequent accuracy of underground prognostications arrived at by instrumental surveying, and considering it to be accomplished only by a sort of magical second-sight, or at least by sagacity, come to expect real second-sight of geologists, and regard such success as merely a matter of course. The results of patient geological investigation are, therefore, apt to be considered mere guesswork, or an ordinary, though mysterious, second-sight; and there is rarely any due appreciation of their real value. A multi-millionaire capitalist, thinking of employing you on a coal-land survey, will ask you half-seriously whether you are able to tell (off-hand, of course) what is 5 fathoms deep below your feet.

But boldly positive and tempting as the declarations of a second-sight man are sometimes made to appear, it is very unsafe, in a case of the least difficulty, to pin your faith upon them without having some satisfactory, rational explanation of their foundation; for they may be grossly misleading. For example, an ore-vein, or a coal-bed, or a set of coal-beds exposed at

one place, may, at a guess, seem, for strong resemblance, or a slight divergence in its course or its character, to be the same or not the same vein, bed, or set of beds as one exposed a quarter or a half a mile away. To take a particular instance, near Schooner pond, on the sea-coast of the principal Cape Breton coal-field, a 3-ft. coal-bed exposed (in 1864) near sea-level appeared possibly to be a different bed from one worked about half a mile away, and again another more than a mile beyond, on the shores of a small headland. An instrumental survey connecting the three points showed them to be perfectly in one straight line and clearly to be upon a single bed, in that region of remarkable uniformity of geological structure. The uniformity, indeed, is so great that the 8-ft. Phalen (or Campbell) coal-bed was, in 1863, successfully opened up on the north side of Big Glace bay, within a few feet of the place indicated by mere instrumental survey from openings about a mile and two-thirds distant on the south side of the bay; and in 1866 was opened further to the northwest at the Caledonia mines by a shaft, where instrumental surveying from the nearest exposure of the bed, three-quarters of a mile distant, indicated that its depth would be 180 ft., and it was found to be 182 ft. But opinions based upon observations without instruments equally suppose the uniformity of the unexposed geological structure to be completely perfect. Yet, even in a case of such uniform, regular structure, it would generally be very unsafe to rely implicitly upon the accuracy of surmises, though they should be made by a wonderfully intelligent man, if they are based merely upon unmapped observations, or generally upon observations mapped without indication of diversities of level—that is, without the topographical map of an instrumental survey; for the mapped position of a high point on a dipping, but not vertical, bed must obviously differ from that of any lower point on the same bed, and in identifying the natural exposures as parts of one bed their elevation, as well as dip, must be taken into account.

Nevertheless, many ungeological and excessively impatient land-owners, or mine-speculators, seem to expect only some such miraculous, yet at the same time accurate, perception of the most hidden facts of a place of supposed mining-promise within half an hour after arriving there by a journey of per-

haps hundreds of miles. Then they are eager to hurry you away and have a report prepared as quickly; or at least to have a preliminary report made without any proper mapping and study of the facts that have been observed. It might be possible in a very plain case, where the beds lie level, or ore-veins are nearly vertical, or where they are evidently quite unworkable; but it is absurd in more complicated cases, with beds of varying dips, and of basin- or saddle-shape, and strongly-curved courses or strikes.

It has already been shown[1] how Lesley played the leading part in bringing topographical mapping to the aid of geological investigations, indicating in 1853 and 1854 the shape of the ground, the hills and valleys, by the more definite and clearer, at that time comparatively novel, contour-lines, instead of the old hachure-lines, with their mountain-ridges looking like caterpillars crawling over the map. The very clearness and definiteness of the contour-lines may seem to require elaborateness and accuracy of instrumental work, but they are also capable of being used advantageously for mere sketching, with indication, of course, that they do not pretend to accuracy.

In 1865 and 1866, a further step forward was taken[2] in indicating the shape of the coal-bed (or other mineral deposit) itself by similar contour-lines, or curves equidistant in level. This, likewise, sometimes seems too precise a method for the uncertain information that may be at hand; but can equally be guarded against being taken as more certain than it really is. The lines are, however, definite, and indicate clearly what is at least supposed to be the geological structure. Like the surface contour-lines, they are a geometrical construction of the shape of the surface to be represented—say, a coal-bed, or other deposit—throughout the area mapped; and the correctness of the structure displayed is severely tested (and perhaps for that very reason the method is less frequently adopted) by its agreement with every known exposure, and by the indication of every possible cross-section, with its series of beds. The method is not only of the greatest value for exhibiting the supposed geological structure, but in the preparation of the map is of yet greater use in working out the most probable

[1] *Trans.*, i., 189–192 (1871–73).
[2] *Trans.*, i., 192 (1871–73); and (J. H. and E. B. Harden) xvi., 290 (1887–88).

structure, by means of numerous trial cross-sections, with the known exposures and their dips and strikes all utilized. Of course, at all the more incomplete stages of the mapping, the tempting second-sight method may be conveniently brought into play, but, of course, too, with uncertain results, which in many cases may be of little value or even seriously misleading. It is true that the complete carrying-out of the instrumental method is laborious and far more time-taking than the field-work, but is well rewarded by the greater certainty of the result. Lesley, the most competent of all judges in the matter, on seeing a few maps made in this way, during his two years' absence in Europe, writes in a letter still extant, June 27, 1868,[3] in warm approval of the method, saying the maps are in "a new style and will probably introduce a new fashion—or rather, I would say, *would* introduce one if there were any well-trained geologists in the country to copy" this "style." But he adds that evidently "it costs enormously in time and brains. I don't object to that myself, you know. And it is the only foundation for a durable reputation." Nearly 20 years later, he, as State geologist, adopted this method of mapping for the great anthracite-survey. The French Geological Survey also has had some of its maps drawn in that way. In Japan many such maps have been made, and a number of them published.

It is true that the instrumental method requires a great deal of time, compared with second-sight. The office-work, if properly done, is so time-taking that our late lamented, able fellow-member, Ellis Clark, formerly assistant on the Pennsylvania State Geological Survey, scarcely exaggerated when he remarked, 20 years ago, that, according to his experience (doubtless without the underground contour-lines), every day of field-work required 10 days of office-work. But a large share of the work, both in the field and in the office, can be done very satisfactorily by a young assistant whose fidelity can be relied on. In the field, he handles the transit, and keeps the survey-notes, with their accompanying sketching, while his chief is free to move about, indicate points to be taken for stations, sketch the topography, and make other observations. In the office, the assistant can, at least, do the plotting, make any needful com-

[3] *Life and Letters of Peter and Susan Lesley*, edited by their daughter, Mary Lesley Ames, vol. ii., p. 74 (1909).

putations, draw the columnar sections and the cross-sections, do the final tracing, the lettering, and the like. Indeed, a somewhat well-trained assistant can do all the field-work, except possibly the interpretation of an exceptionally knotty point here and there; and so in the office he could, with a little guidance, do all the work, except the decision of the most important questions. Lesley so worked, especially in regard to the field, which he in some cases would hardly visit at all, sometimes not at all; but he was not averse to the office drudgery, at which he was very adroit, and which he found agreeably reposeful and invigorating, as well as highly conducive to a thorough digestion of the elementary facts.

Of course, instrumental surveying does not always need to be done with instruments of the highest precision. The leveling does not need to be so exact that a polygon will close with an error of only 0.01 ft. Leveling with a pocket-level that closes within a foot or so may be satisfactory, if there be checks to prevent such errors from accumulating, for the rock-beds themselves are more variable than that in thickness. Aneroid leveling, though still less exact, may advantageously be used, if frequently checked by more exact work. For the horizontal work, the large transit-compass is generally precise enough; and even the small prismatic compass is useful, far beyond mere sketching or guessing. Stadia measurement, if without too long sights, is at least as good as chaining; and careful pacing is much better than guessing at distances, and was found by Prof. H. S. Munroe, 35 years ago, in extensive tests of the work of four men, to average in error only about 0.5 per cent. These rough methods are, at any rate, a great improvement upon sketching alone, or second-sight and guess-work, which, to be sure, may be much better than nothing, according to the personal skill and eye-sight of the observer, though apt at times to be grossly erroneous and misleading in spite of the utmost sagacity.

An example of the advantage of merely rough surveying is to be seen in the Pennsylvania State Geological Survey map of the New Red of Bucks and Montgomery counties (1893), where the probable place of outcrop of two important beds of brownstone, valuable for building-purposes, is indicated by two crooked lines running westward many miles through the coun-

try from large quarries near the Delaware. The crooks and bends in the lines may by the ignorant be supposed to be mere fancy work, but were drawn with care according to the height of the ground and the direction and steepness of the dips, turning northward in the low ground, and rising southward in the higher land. Of course, with the unsatisfactory means at hand for so hasty a survey, no very great accuracy could be expected from such indications; and, indeed, nobody apparently put enough faith in them, if at all understanding their meaning, to try with their help to open up the valuable building-stone at any point where it would be convenient to quarry it. Nevertheless, some years after the publication of the map, a quarry in the stone, seemingly discovered by chance, was opened on the marked outcrop at Grenoble, 5 miles west of the nearest old quarry; and another, in the same way, 5 miles still further west, at the trolley power-house. Successes like that in geological mapping are apt, if noticed in any way, to be taken as a matter of course; as if it did not, after all, require any particular intelligence, or common sense, to adopt the so universally neglected instrumental, topographical method, albeit in a field that had for three-quarters of a century been geologically under the unsatisfactory sway of second-sight (or closely kindred) sorts of investigation. Or the successes are apt to be taken as not needing any special care or skill after once adopting a plan so simple, to be sure, in principle, yet so capable of being negligently and inaccurately carried out. In fact, not only ungeological men, but beginners in geology, seem to suppose that such precise indications are altogether cases practically of second-sight, which a geologist is to be expected as a matter of course to possess in a high degree of accuracy—at least as high as any other geologist has. They suppose that "they all do it," and imagine that all that is needed in their own statements of opinion is to "be bold, be bold, be bold," forgetting that they must "be not too bold."

A neglect to complete topographical mapping and to make neighboring cross-sections that would properly take into account the varying course of the rock-beds and their elevation, did in one coal-field lead a highly-sagacious observer to mistake the true identity of the coal-beds and their probable place of outcrop, and to dig in one place a long drift, in another, a deep shaft,

in the vain expectation of finding certain coal-beds at points where more thorough mapping and cross-sections later showed their existence to be clearly quite impossible. A beginner's youthful, but readily pardonable and not unattractive enthusiasm, at another point in the same field, led to his having somewhat extensive fruitless digging done where later the completion of the map showed for the whole neighborhood that such digging would plainly be useless. In another part of the same field certain dips seemed by the second-sight method to indicate great irregularity, as if from numerous and extensive faults; yet a patient investigation with mapping made it clear that no such great disturbance probably existed thereabouts, and that the coal-beds were there likely to be in satisfactorily workable condition. Again, in the same field, before the mapping was completed, two whole sets, each of three or four workable coal-beds, were quite naturally (by second-sight method) supposed to be but one set, though the map later made it sure that they were two distinct sets of beds, overlying one another, and thereby giving to the field a far greater amount of workable coal than had previously been suspected. In such a case, a hasty, yet not incautious, "preliminary report," without waiting for the completion and study of the map and of many trial cross-sections, could hardly fail to be quite erroneous and misleading.

There are countless instances where such geometrical construction of the geological structure has corrected second-sight guesses, and successfully guided the opening-up of coal-beds, visibly and far more satisfactorily than drilling deep holes in the way so fascinating and costly to many men unfamiliar with the capabilities of instrumental surveying. But why multiply special citations?

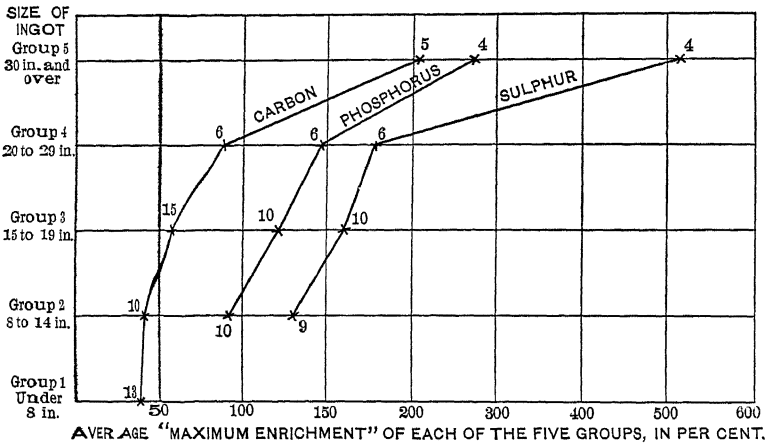
It is already fully evident that second-sight, tempting as it is for its hare-like celerity, cannot, for certainty of arrival at a satisfactory goal, in the least compare with the invaluable, surer, steady-going, though, if you please, more tortoise-like, process of instrumental surveying. The tortoise is the favorite Japanese emblem of great longevity; and as such might well be applied to the long-lasting useful results of this too-much-neglected method.

The Influence of Ingot-Size on the Degree of Segregation in Steel Ingots.

BY HENRY M. HOWE, NEW YORK, N. Y.*

(Spokane Meeting, September, 1909.)

THE natural effect of large ingot-size should be to increase segregation. I have previously pointed[1] to the excessive segregation in many large ingots as tending to confirm this, but I have shown that in case of ingots less than 16 in. square this expected effect of ingot-size is liable to masked by that of



NOTE.—The "maximum enrichment" of each ingot, *i.e.*, the excess of the richest spot over the average of the whole ingot, is first calculated in percentage of that average. The average of the maximum enrichment of the several ingots of a given group is the abscissa in Fig. 1. The number beside each spot tells the number of cases which that spot represents.

FIG. 1.—INFLUENCE OF INGOT-SIZE ON MAXIMUM ENRICHMENT IN STEEL INGOTS.

other variables. Under these conditions we should expect that, if large ingot-size really does tend to increase segregation, this effect would be shown by taking the average of large numbers of cases, so that the effects of these other variables might off-set and cancel each other.

* Professor of Metallurgy in Columbia University, New York, N. Y.

[1] *Engineering and Mining Journal*, vol. lxxxiv., No. 22, p. 1015 (Nov. 30, 1907).

A. Cavity.

FIG. 2.—SECTIONAL VIEW OF INGOT. FULL SIZE.

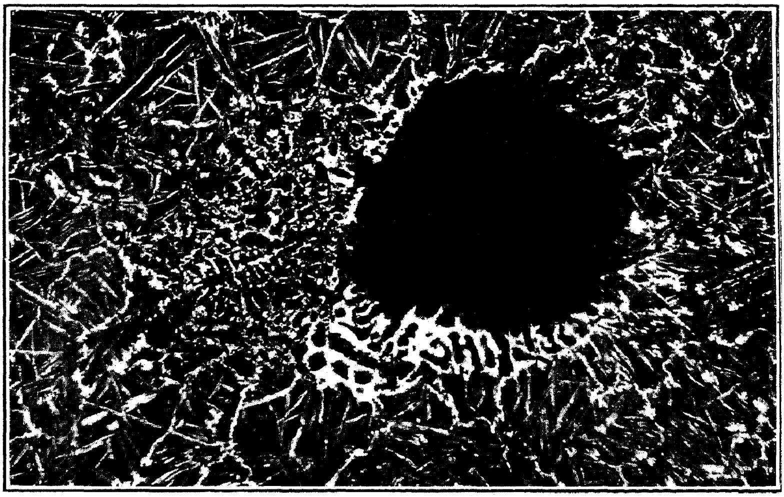
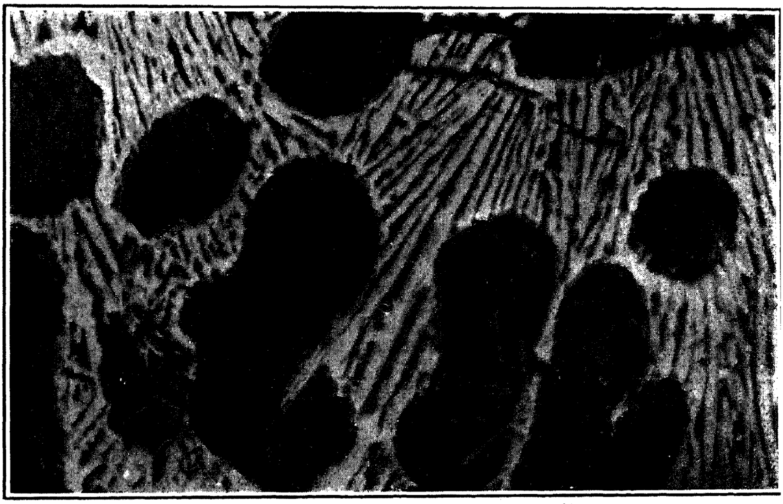


FIG. 3.—CROSS-SECTION THROUGH THE SMALL CENTRAL CAVITY. MAGNIFICATION, 60/1.



B

Fig. 4.—Cross-Section Through the Small Central Cavity. Magnification, about 130/1.

That the degree of enrichment does increase with ingot-size when thus studied is shown by Fig. 1, which represents the average degree of enrichment in 49 different ingots, divided into five classes, according to their size. This figure also brings out prominently the fact that the enrichment in sulphur is greater than that in phosphorus, and that in phosphorus greater than that in carbon. The detailed data on which this figure is based I hope to publish soon. This figure further tends to show that the effect of ingot-size is relatively slight until the thickness of the ingot reaches something like 20 in., but that with further increase of size the enrichment increases more rapidly. This diagram is based on the enrichment at the richest point found in each ingot.

That even very small ingots may be greatly enriched by segregation is shown by Figs. 3, 4, and 5, which represent the microstructure of the neighborhood of a small cavity in the upper part of the axis of a small test-ingot, Fig. 2, only about $\frac{15}{16}$ in. (or 0.94 in.) wide at its widest part and about 5 in. long. It is not necessary to discuss here whether this is a true case of axial segregation or not. My present purpose is to put this interesting case on record. Fig. 5 shows that some of the metal had been enriched in carbon so much as to have turned into white cast-iron, with somewhere about 3 per cent. of carbon. Indeed, the eutectic areas must contain more than 4.3 per cent. of carbon.

The ingot itself is a little acid open-hearth test-ingot, which, after it had sunk to a moderate red heat, was quenched in water as a matter of convenience. Hence the martensitic structure. The metal from the outer part of this ingot contained 1.08 per cent. of carbon by combustion, as determined by J. O. Handy, of the Pittsburg Testing Laboratory, so that the enrichment even in this minute ingot is not far from fourfold.

A New Separator for the Removal of Slate from Coal.

BY W. S. AYRES, HAZLETON, PA.

(Spokane Meeting, September, 1909.)

[SECRETARY'S NOTE.—At the Spokane meeting of the Institute, in discussion of President Brunton's address on "Modern Progress in Mining and Metallurgy in the Western United States," and at the request of members present, Mr. Ayres gave an oral account of his new separator, which is here published as an independent paper, partly because of its inherent importance and partly because it describes an improvement which did not originate in the Western United States, and therefore does not fall, strictly speaking, under the title of President Brunton's address.—R. W. R.]

A BRIEF history of the growth of the anthracite-coal preparation will give a better view-point from which to judge the present problem of separating slate from coal.

At the beginning of the commercial value of anthracite, 70 years ago, only the pure portions, or "splits," of the veins were mined and shipped to market, and without any preparation or screening other than the selection, while loading, of the glassy lumps, and the rejection of the fine material and the slate that had strayed accidentally into the coal. The next step was the crushing or breaking of the coal (hence the name "breaker" as applied to the preparation-building), and the sizing of it by means of bars or revolving screens. This stage of its development marked the advent of the "breaker-boy" as a slate-picker, with his ever-increasing capriciousness.

As the richer veins or "splits" became exhausted and the market demanded a still greater output, the less-pure "splits" and the thinner veins were utilized to produce the coal. Carrying as they do a far greater percentage of impurities, particularly when removing the pillars, it became necessary to build new and better equipped preparation-plants. Finally, we are now at the highest stage of complication yet known to the art of coal-preparation. We are dealing with varying specific gravities, frictional difference, hardness, structure, and form in the pieces of coal and slate coming from many different veins, and

all mixed together in different proportions. The treatment of each vein separately is, of course, impossible because of the size of the plant required. The re-treating of the refuse-banks from the early mining-operations is, however, generally done in an individual plant, termed a "washery," separate from that used for fresh-mined coal.

These complications in handling the material have brought forward several types of jigs and mechanical pickers, all of which are more or less wasteful of the coal.

A very careful study during the past 15 years of the conservation of coal after it has been delivered from the mine to the preparation-plant, or "breaker," has led me to the devising of means to prevent the very great losses sustained. These losses consist of undue chipping of the coal in the process of crushing, in the process of screening or sizing, in the jigs and other separating-machinery, and in the conveying-chutes, or "telegraphs," as they are called. Fundamentally, every impact destroys values by chipping from the larger pieces, which have the greatest value, very small particles which are practically valueless. These losses range from 1 to 2 per cent. in a right angle bend in a straight chute, or "telegraph," to 20 per cent. in a jig. In the preparation- or slate-picking machinery alone the losses range usually from 5 to 20 per cent. The picking of slate by hand is very wasteful also. The average boy throws out about as much coal as he does slate, and much more on dark days and at night.

In the construction of the coal-breaker at the Cranberry mine, at Hazleton, Pa., in 1896, I designed and introduced for the first time the continuous spiral chute, which is fully described in my paper, The New Breaker at Cranberry Coal-Mine. [1] This chute, having pitches determined by experiment for each size of coal, delivers the coal from the separating-machines to the pockets at a very moderate speed, and with a very decidedly smaller amount of loss in chippings than the ordinary straight chute. The saving is about 2.5 per cent. This chute is now quite extensively used throughout the region.

Material losses in the crushing-rolls and screens have been greatly reduced by improved types of these machines. In

---

[1] *Trans.*, xxviii., 293 (1898).

1896 I gave to a large manufacturing concern some data showing that the rolls should be at least 48 in. in diameter.

The greatest losses, however, are in the separating-machinery. All jigs are destructive because of the 80, more or less, impulses given the material per minute, thus continually grinding from the coal small particles which are valueless. All mechanical pickers heretofore designed are also very destructive, because the coal must sustain severe impacts in passing through them, and consequently create very great losses.

It is with a view to avoiding these losses that I have made many exhaustive tests at my testing-plant.

There are only two handles, so to speak, known to the art of separating slate from coal, by which we can mechanically take hold of the problem; one is difference in specific gravity, applied in jigging, and the other is frictional difference, or difference in the angle of repose, applied in frictional separators.

A somewhat better understanding of it may be had from a study of the material itself. Much of the material as it comes from the screens is composed of the following different forms of coal and slate, classified for convenience into six groups:

(1) Glassy fractured coal, usually cubical in form.

(2) Flat coal, some pieces having slate faces.

(3) Bone (interlaminated coal and slate), usually flat, and either coal-faced or slate-faced.

(4) Flat slate, from 0.25 to 0.5 in. thick.

(5) Pure slate with coal faces, approximately cubical in form.

(6) Slate and rock, heavy, and cubical in form.

The jig, in addition to the fault of seriously abrading the coal, does not effect a good separation when working on material composed of pieces having such widely-differing forms. In the coal-discharge is found a most unsatisfactory and aggravating mixture of slate and coal. Groups (1) and (2) predominate, but with them is found a large percentage of group (4)—flat slate—and a considerable quantity of groups (5) and (6)—pure slate and rock. In the slate-discharge, on the other hand, are found groups (5) and (6) predominating, with a high percentage of group (1)—glassy fractured cubical coal. The fact that not a single piece of coal taken from the slate-discharge has ever been found with a specific gravity greater than even the lightest piece of slate, is conclusive. Therefore the old and

oft-repeated explanation of this erratic action of jigs, viz., "that the specific gravity of some of the coal is greater than that of the slate," becomes a myth.

It is plain, therefore, that a jig is inefficient with this class of material, which to-day constitutes the greater part of that brought to the "breaker," and that the form-difference of the

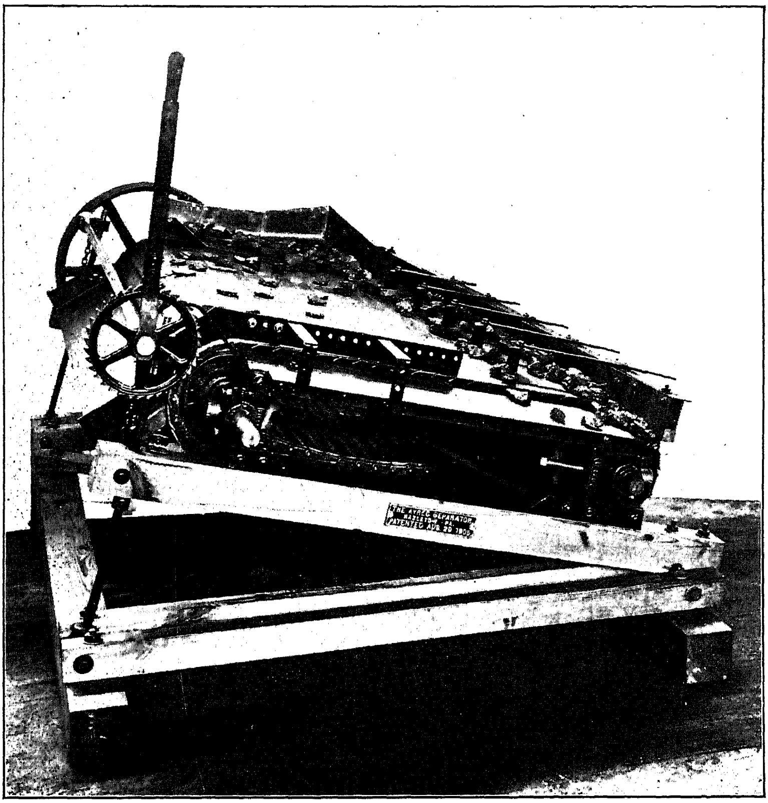


FIG. 1.—THE AYRES SEPARATOR.

pieces has a dominating influence over the process of separation. The cause of this erratic action has been so well defined by classifying the pieces composing this class of material into form-groups and then determining individually their weights, areas exposed to the impulses of the jig, and their specific gravities, that the wonder is that the jig effects any separation at all. In practice, when certain groups predominate there is really no separation. The cause lies in the fact that the area exposed to

the impulses of the water does not have a uniform ratio to the cubic content of the pieces. Consequently, a piece of flat slate is sure to be lifted into the coal-zone, and, on the other hand, a cubical piece of coal can be dropped into the slate-zone. In fact, because of the shifting positions of these more or less flat or elongated pieces, from edgewise to flatwise to the impulses of the water, no distinct and progressive separation-zones are established in the jig: only a continuous mixing is the result. The ideal conditions for a jig are that the material shall be composed of spheres of the same diameter, and that the coal shall have a specific gravity that is lighter than that of the slate.

It is also well established that group (2)—flat coal with slate faces—and group (5)—pure slate with coal faces—cannot be separated by any frictional separating-machine: in fact, the jig is the only separating-device that will make the separation.

In view of the foregoing difficulties I have designed a separator that would. at least, meet the greatest of them. The chief objects in its design were (1) the avoidance of all impacts and the consequent wasteful chipping of the coal; (2) the doing away as far as possible with hand-picking; (3) the removing of flat slate without wasting the flat coal; (4) the exposing of the operation to view, so that the exact action of the machine could be observed at all times at a glance; and (5) the ability to adjust the machine while in operation.

Fig. 1 clearly shows the construction and operation of the machine.

The traveling separating-belt, mounted on two shafts, is made to move upwardly on its upper run by means of the drive-pulley. The belt, made of metal slats attached to a specially-designed link-belt, is inclined forwardly as well as transversely, and at such angles as are suitable for the proper separation of the material to be treated. Its transverse inclination is adjustable, while running, by means of the lever and ratchet-wheel.

The material is fed in a continuous stream upon the pan at the farther and higher end of the machine through a properly-constructed feed-chute. As it slides off of the pan the upwardly-moving belt immediately spreads the material out into a well-spaced stream, so that the coal may slide down against the guide and off of the lower right-hand corner of the machine,

while the slate, adhering to the belt, may move upwardly to the left-hand side and thus out of the forwardly-moving stream of coal.

The coal is conveyed away from the machine in any desired direction by a suitable chute, and the slate in like manner is collected along the slate-apron at the left in a similar chute and disposed of as desired.

The capacity of the machine, which varies somewhat with the nature of the material, but chiefly with the size of the coal, ranges from 5 tons per hour on pea-size (through a 0.75- and over a 0.5-in. mesh), to 25 tons per hour on steamboat-size (through a 6- and over a 4.5-in. mesh).

The operation is entirely open to view.

The saving by preventing the loss in chipping, which, as already stated, ranges in other separators from 5 to 20 per cent. of the coal treated, is enormous. The chippings on this new separator amount, on an average, to less than 0.5 per cent. On the low basis of 5 per cent. saved, the amount would be $18,750 on every 100,000 tons of prepared sizes shipped. One installation of 10 machines, now in operation for more than a year, shows a gain of 19.5 per cent. in the prepared sizes, with an output of 11,500 tons per month, or a gain of $4,152 per month. In addition to the gain by the prevention of chippings, a saving in labor of $500 per month has been effected.

The gain by the prevention of chippings is far greater than the saving in labor; in this case it is more than eight times greater.

The separator has the ability to handle material carrying a widely-varying percentage of slate, ranging from 10 to 90 per cent., and to give a practically uniform product. It is these "peak-loads" of slate, if I may borrow an electrical term, that have universally caused trouble in former methods. It also has the ability to remove the flat slate without removing the flat coal.

The many installations on steamboat-size (which is usually hand-picked) show a very great saving in labor, each machine doing the work of from 5 to 12 men, besides giving a decidedly more uniformly clean product, which, when crushed to prepared sizes, as is now almost universally done, reduces the labor still further on the preparation after crushing.

It is not possible in all cases to do away entirely with hand-picking by the use of the separator, because of the occasional presence of pieces of slate having coal faces, and pieces of coal having slate faces; but instead of having, as in one case, 32 men hand-picking on steamboat-size, now 4 of these separators and 12 men do the work. The period of time required to pay for the equipment by its actual saving ranges from 10 to 80 days.

In treating such material as has been described in connection with the jig, it has been found most effective to use a three-stage process. The first process to be a set of these separators, turning direct to the pocket, without appreciable loss from chippings, 60 to 75 per cent. of the total coal in the material, which includes all the glassy fractured cubical coal—group (1)—and all of the flat coal not having slate faces. The second process to be likewise a set of these separators so adjusted as to remove all the flat slate—group (4)—and all pure slate and rock—group (6). With the foregoing mixture of forms trimmed, so to speak, in this way, there remain group (2), flat coal with slate faces; group (3), bone; and group (5), pure slate with coal faces. Passing this product, which contains only from 25 to 40 per cent. of the total coal, to a jig as the third process, a very satisfactory separation can be made.

By this three-stage process the losses have been reduced to a minimum, or, to put it the other way, the gains as shown in the installation of the 10 machines previously mentioned have been as high as 19.5 per cent. in the prepared sizes.

*

The Barometric and Temperature Conditions at the Time of Dust-Explosions in the Appalachian Coal-Mines.

BY N. H. MANNAKEE, WILLIAMSON, W. VA.

(Spokane Meeting, September, 1909.)

SINCE the publication of the paper of Mr. Scholz, The Effect of Humidity on Mine-Explosions,[1] I have undertaken a study of the meager available data of barometric and temperature conditions at times of mine-explosions, and have arranged these data in a manner which may be of interest to other students of mine-explosions.

This study covers the period from 1898 to 1909, during which an ever-increasing percentage of collieries has been ventilated by fans giving air-currents of high efficiency. Previous to 1898 this percentage was smaller than at any time since. The various States have become, from time to time, more exacting in their demands upon poorly-ventilated mines. This general improvement in ventilation throughout the Appalachian field has undoubtedly brought about a condition which is annually the cause, or a very important attendant, of the high percentage of fatalities in mine-explosions, through the daily passage of from hundreds to thousands of gallons of water, invisibly suspended in the air, carried out of the mines by the ventilating-currents during the existence of certain temperature- and humidity-conditions.

There is a wide-spread opinion that these conditions exist from November 1 to March 31; but it is more likely that they have a much wider range, which varies from year to year. Certain years show an increase or decrease of rain-fall from the mean; the daily temperature may fall far below or rise far above the mean established by a record of a period of years; and the daily humidity varies likewise. If mine-explosions are influenced by a lack of moisture in the mines, certainly it may be expected that the influence will be felt at times when the

[1] *Trans.*, xxxix., 328 to 336 (1909).

moisture falls below the safety-point, and this relative condition may appear from year to year in the months of September and October, and April and May. Since the carrying-power of air for water-vapor diminishes as the temperature falls, there will be experienced in general a reduction of moisture in the mines the moment the intake temperature falls below that of the return-air. In proportion to the extent and continuance of this difference will the mine become drier, unless some artificial means is used to supply moisture up to the saturation-point of the upcast current.

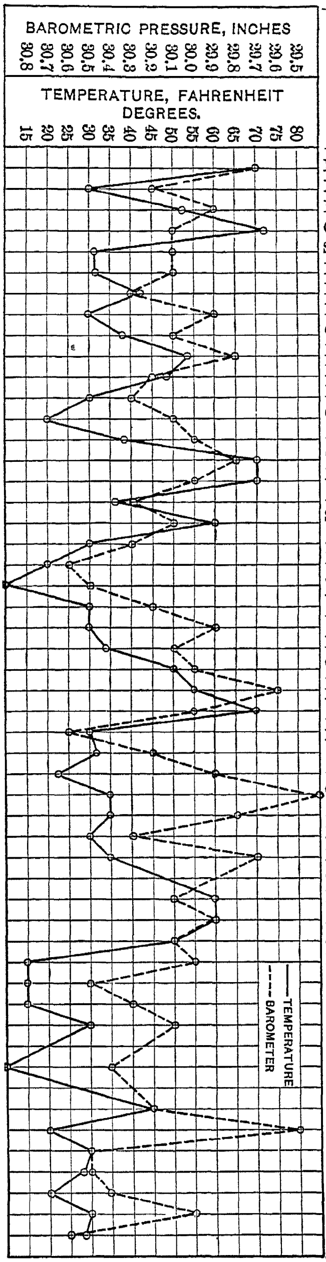
This robbing of a mine of moisture is dangerous if carried to a certain point, and this point varies under different conditions in the same locality; but the exact point at which it should be placed remains to be determined. The presence of dust in a mine, even in one portion only, has repeatedly been the cause of an explosion; and the violence of the explosion is determined by the extent of the dusty territory. The dust-explosion requires certain attendant circumstances other than the mere presence of dust, such as the initial temperature produced by a blown-out shot, the explosion of powder, etc. If these conditions could be eliminated, there would be no dust-explosions (except through the bare possibility of ignition by spontaneous combustion). They are mostly caused by ignorant, careless, or untrained miners, and we shall probably always have a certain percentage of this class of labor around the mines. So long as that is the case, the occasional direful consequences will be reaped.

I believe the real solution of the problem is, so to control the temperature and humidity of the mine-air that a dust-explosion will be impossible. I regard such control as practicable.

A careful study of the conditions can only be made after the systematic collection of data. The data herewith presented are not exact; but, so far as I can ascertain, they are sufficiently close to the truth to justify the arguments drawn therefrom. Some of them are not new, but are presented in order to make comparisons which I have not seen elsewhere.

Table I. is a complete list of mine-explosions occurring in the Appalachian coal-field from 1898 to 1909 in which five or more fatalities occurred, and which were caused by dust or gas, or both. This list is made up from the one published by

TABLE I.—*Mine-Explosions Supposed to be Caused by Gas, Dust, or Dust and Gas, Causing Five or More Fatalities, Officially Reported in the Appalachian Coal-Field, North America,* 1898–1909 *Inclusive.*



Location.	Date.	Number of Fatalities.
Brownsville, Pa	Sept. 23,'98	8
Mahanoy City, Pa	Mar. 9, '99	8
Dominion Colliery No. 4, Nova Scotia	June 16,'99	11
Grindstone, Pa	July 24, '99	5
Sumner, Pa	Dec. 23, '99	20
Red Ash, W. Va	Mar. 6, '00	46
Berryburg, W. Va	Nov. 2, '00	15
Buck Mt. Colliery, Pa	Nov. 9, '00	7
Chatham, W. Va	May 15, '01	10
Richland, Tenn	May 20, '01	20
Port Royal, Pa	June 10,'01	19
Buttonwood, Pa	Oct. 25, '01	6
Catsburg, Pa	Mar. 6, '02	5
Dayton, Tenn	Mar. 31, '02	16
Coal Creek (Fraterville), Tenn	May 19, '02	184
Rolling Mill Mine, Johnstown, Pa	July 10, '02	112
Algoma, W. Va	Sept. 16,'02	17
Stafford, W. Va	Sept. 22,'02	6
Luke Fidler, Pa	Nov. 29,'02	7
Ferguson, Pa	Nov. 21,'03	17
Cheswick (Harwick Mine), Pa	Jan. 25, '04	176
Virginia City, Ala	Feb. 20, '05	111
Wilcox, W. Va	Feb. 26, '05	7
Red Ash, W. Va	Mar. 18,'05	24
Cabin Creek, W. Va	Apr. 20, '05	6
Dubois, Pa	Apr. 27, '05	13
Vivian, W. Va	July 5, '05	5
Hazel Kirk, Pa	Oct. 29, '05	5
Vivian, W. Va	Nov. 4, '05	7
Bentleyville, Pa	Nov. 15, '05	6
Coaldale, W. Va	Jan. 4, '06	22
Detroit, W. Va	Jan. 18, '06	18
Parrall, W. Va	Feb. 8, '06	27
Piper, Ala	Feb. 27, '06	12
Century, W. Va.[a]	Mar. 22, '06	23
Huger, W. Va	July 19, '06	5
Pocahontas, Va	Oct. 3, '06	36
Johnstown, Pa	Oct. 24, '06	7
Penco, W. Va	Jan. 26, '07	12
Fayetteville, W. Va	Jan. 29, '07	85
Elkins, W. Va	Feb. 4, '07	25
Whipple, W. Va	May 1, '07	16
Naomi, Pa.[a]	Dec. '07	35
Monongah, W. Va	Dec. 6, '07	350
Darr, Pa.[a]	Dec. '07	239
Yolande, Ala	Dec. 16, '07	16
Port Hood, Nova Scotia	Feb. 10, '08	10
South Carrollton, Ky	Feb. 10, '08	9
Marianna, Pa	Nov. 28,'08	154
Lick Branch, W. Va	Dec. 28, '08	50
Lick Branch, W. Va	Jan. 12, '09	48
Birmingham (No. 2 Short Ck.), Ala	Feb. 2, '09	18

Total fatalities, 2,126

[a] No data. Temperature assumed at 32°.

H. N. Eavenson,[2] and from certain other data collected personally to bring it to date. Opposite this chronologically-arranged list are recorded in curves the approximate temperature and pressure which obtained at the time of each explosion. These data were collected at the Weather Bureau at Washington, D. C., and while in most cases not absolutely accurate, may be taken as approximately so. The curve shows at a glance the range of explosion-temperatures, the highest recorded being 70° and the lowest 10° F. The barometric curve shows the range of pressure.

There are, no doubt, a number of the explosions listed in Table I. in which both gas and dust figure as causes. The questions concerning gas and dust combined are undergoing experimental investigation at the U. S. Geological Survey Technologic Bureau at Pittsburg, Pa., and much interesting information is being secured. But the question, how to control temperature and humidity of intake mine-air, is receiving minor consideration.

TABLE II.—*Barometric Condition at Times of Explosion.*

Explosions.	Number.	Per Cent.
In high-pressure areas,	23	46.94
In mean-pressure areas,	18	36.73
In low-pressure areas,	8	16.33
Total,	49	100.00

Table II. shows the barometric condition at the time of explosions. The daily weather-maps, issued by the United States Weather Bureau, show lines of equal barometric pressure; and certain points where explosions occurred were located in high-pressure or low-pressure areas, or occupied a mean between high and low pressure. Table II. shows that out of 49 explosions for which data were available, 23, or 46.94 per cent., took place in high-pressure areas. The number of explosions which occurred in mean-pressure areas was 18 out of 49, or 36.73 per cent. The number of explosions occurring in low-pressure areas was 8 out of 49, or 16.33 per cent. This indicates that the pressure is generally likely to be high at the time of an explosion. Dry air is heavier than damp air, hence dry air indicates high barometer.

[2] This volume, p. 837.

Table III. shows the number and percentage of explosions at certain temperatures and the number and per cent. of fatalities.

TABLE III.—*Explosions at Various Temperatures, and Number of Fatalities, Reduced to Percentage.*

External Temperature. At or Below.	Number.	Per Cent.	Number of Fatalities.	Per Cent.	Number of Fatalities per Explosion.
32° F., . . .	29	55.77	1,539	72.39	53.07
40° F., . . .	36	69.23	1,658	77.98	46.06
50° F., . . .	41	78.85	1,721	80.95	41.9
60° F., . . .	47	90.39	1,812	85.23	38.6
Above 60° F., . .	5	9.61	314	14.77	62.8
Total explosions considered, . .	52	100.00	2,126	100.00	40.9

Particular attention should be given to the explosions occurring above 60° F. Here are 5 explosions out of 52, or 9.61 per cent., and in these explosions 314 out of a total of 2,126 fatalities, or 14.77 per cent. It is my opinion that these 5 explosions, occurring above 60° F., were strictly gas-explosions. Thus far I have not had access to the printed reports of these cases, but I feel that it is not probable that dust caused them in any way. There is also a certain percentage of the explosions which occurred at 60° F. and below which are distinctly gas-explosions and in which dust plays no important part; and these should not be included in the list. There are well-defined means of regulating purely gaseous mines; and such accidents should not be included in a careful study of dust (and gas-and-dust) explosions. Table III. is, therefore, not a fair *résumé* of explosions which have in whole or in part been caused by the presence of dust. It will be possible to rearrange this table later by eliminating all explosions caused by gas alone.

Table IV., reproduced from the paper of H. N. Eavenson,[3] shows that during the winter months water is removed from the mines by the ventilating-current at the rate of from 2,777 to 16,694 gal. per 24 hr.; and that in the summer months it is deposited in the same mines at the rate of from 2,978 to 8,088 gal. per 24 hr. These figures are probably the extremes; but I think that June, July, and early August will show

[3] This volume, p. 835.

TABLE IV.—*Humidity-Tests at Various Mines in Southern West Virginia.*[4]

Mine No.	Date.	Outside Air.			Mine Air.			Aqueous Vapor Removed.	Quantity of Air in Circulation.	Amount of Water Removed from Mine.			
		Tempera-ture.	Saturation.	Aqueous Vapor.	Tempera-ture.	Saturation.	Aqueous Vapor.			Per 24 Hr.		Per Minute.	
		F.°	Per Cent.	Gr. Cu.Ft	F.°	Per Cent.	Gr. per Cu. Ft.	Gr. Cu. Ft. Air.	Cu. Ft. per Min	Gal.	Short Tons.	Gal.	Lb.
1....	1/24-08	21.8	44	0.591	53.5	97	4.467	3.876	130,000	12,428	51.8	8.6	72.0
1....	2/17-08	29.5	52	0.985	53.8	85	3.955	2.970	128,000	9,377	39.1	6.5	54.3
1....	2/18-08	40.5	44	1.277	53.5	84	3.868	2.591	128,000	8,181	34.1	5.7	47.4
1....	2/19-08	41.0	78	2.305	53.5	85	3.914	1.609	128,000	5,080	21.2	3.5	29.4
1....	2/20-08	28.3	66	1.186	53.3	82	3.751	2.565	128,000	8,098	33.8	5.6	46.9
2....	2/18-08	22.0	79	1.070	53.0	97	4.390	3.320	144,000	11,792	49.2	8.2	68.3
2....	2/19-08	41.8	70	2.130	52.8	98	4.405	2.275	144,000	8,081	33.7	5.6	46.8
2....	2/20-08	25.7	75	1.201	53.0	97	4.390	3.189	144,000	11,326	47.2	7.9	65.6
2....	2/21-08	27.5	76	1.319	52.3	97	4.286	2.967	144,000	10,558	43.9	7.3	61.0
2....	2/22-08	30.0	47	0.910	52.5	97	4.316	3.406	144,000	12,097	50.2	8.4	70.0
2....	2/24-08	32.8	56	1.215	52.7	98	4.390	3.175	144,000	11,277	47.0	7.8	65.3
2....	2/25-08	37.6	72	1.877	52.5	98	4.360	2.483	144,000	8,820	36.8	6.1	51.1
2....	2/26-08	35.5	83	2.156	52.6	99	4.420	2.264	144,000	8,041	33.5	5.6	46.6
6....	2/21-08	38.0	43	1.137	52.8	94	4.225	3.088	200,000	15,234	63.5	10.6	88.2
6....	2/22-08	24.5	68	1.031	50.5	97	4.024	2.993	200,000	14,765	61.6	10.2	85.5
6....	2/24-08	39.0	35	0.961	52.7	97	4.345	3.384	200,000	16,694	69.6	11.6	96.7
6....	2/25-08	29.3	56	1.051	52.4	97	4.301	3.250	200,000	16,033	66.9	9.1	92.9
2A.	Jan., 08	41.0	92	2.719	59.0	94	5.222	2.503	105,000	6,483	27.0	4.5	37.5
2A.	Feb., 08	37.0	91	2.321	58.0	94	5.048	2.727	106,000	7,130	29.7	4.9	41.3
1A.	Mar., 08	54.0	76	3.561	56.0	93	4.665	1.104	102,000	2,777	11.6	1.9	16.1
								Aqueous Vapor Deposited.		Amount of Water Deposited in Mine.			
								Gr. Cu. Ft. Air.					
6....	8/19-08	83.5	57	6.938	60.0	99	5.687	1.251	150,000	4,629	19.3	3.2	26.8
6....	8/20-08	71.0	80	6.592	60.0	99	5.687	0.905	150,000	3,348	14.0	2.3	19.4
6....	8/22-08	72.0	93	7.912	60.2	99	5.726	2.186	150,000	8,088	33.7	5.6	46.8
4....	8/24-08	80.0	58	6.342	59.2	99	5.437	0.805	150,000	2,978	12.4	2.1	17.2

larger quantities of water deposited. There is a gradual passage, step by step, from the condition of robbing the mines of moisture in winter to the deposition of moisture in summer; and there is a time at which the ventilating-current neither gives nor takes moisture in traveling through the mine. I believe that this balanced condition occurs in southern West Virginia from time to time, between the last of March and the middle of May, alternating between short periods when the mines are robbed of moisture and periods when they receive moisture from the ventilating-current. These fluctuations occur until about the middle of May, after which the summer condition very likely prevails until the middle of September, when we again experience the variation between excess and scarcity

[4] This volume, p. 836.

of moisture. Then, in the last part of October or the early part of November, the winter condition is installed again, and the mines give up their daily quota of water to the ventilating-current. This opinion is by no means established by complete evidence; yet a comparison of the daily-temperature curves at a particular place tends to confirm it. Since the temperature of the intake air indicates to a large extent the quantity of moisture which it carries, we can assume for practical purposes that when the outside temperature is below the mine-temperature the mine will give up moisture to the ventilating-current.

Table IV. shows that the degree of saturation of mine-air varies from 82 to 99, while that of the outside air varies from 35 to 93 per cent. The percentage of saturation of the outside air in winter is evidently less than in summer, and the same condition exists in regard to the mine-air, but is not so pronounced. Considering the saturation of the outside air at different temperatures, Table IV. indicates that the lower the temperature the smaller the percentage of saturation. And since the carrying-power of the air diminishes as the temperature falls, it would be expected that the quantity of moisture carried out of a mine in 24 hr. will increase in greater ratio than the difference in temperature between the intake and the effluent air. In other words, there are two controlling conditions:

1. Lower temperature usually means lower percentage of saturation; and

2. Lower temperature indicates lower carrying-power for moisture.

On the other hand, increase of temperature indicates a higher percentage of saturation, and higher temperature indicates higher carrying-power for moisture. Thus it would seem that when the temperature of the intake air is 5° higher than the temperature of the effluent air, a certain quantity of water would be deposited in the mine, and when the temperature of the intake air is 5° lower than that of the effluent air a much larger quantity of water would be carried out of the mine. It naturally follows that at those times of the year when the outside temperature falls several degrees below the inside temperature the mines will become drier. The greater this difference in temperature of intake air and effluent air, and the

longer this condition exists, the more pronounced will become this dry condition. All of those mines which become dusty and which are liable to be dangerous on this account have a certain safety-point which lies somewhere between the extremes pointed out above. There are no definite records to show at what temperature certain mines begin to become dry, or when they will become wet. Yet these two questions are on the very threshold of the inquiry.

In order to facilitate the thorough investigation of this phase of mine-explosions, I recommend that the following data be recorded daily at each mine employing a certain number of men in the Appalachian coal-field:

1. Temperature of outside air (continuous record).
2. Temperature of inside air (continuous record).
3. Humidity of outside air (continuous record or several readings daily).
4. Humidity of inside air (continuous record or several readings daily).
5. Volume of air in circulation.
6. Number of hours in circulation.

These data, accumulated through a period of months, preferably years, would, I believe, shed light on this serious problem and would bring us nearer to a knowledge of the effect of humidity in mine-explosions. By this means of collecting data the fluctuation in the moisture-content of the mine-air could be ascertained, and the amount of water carried out of any particular mine or deposited therein by the ventilating-current for any period could be determined. Then, when dust-explosions occurred, reference could be made to the daily record of conditions at the mines, and from this starting-point the probable cause could be sought.

So long as it is not known at just what point of relative dryness the condition of a mine becomes dangerous, we shall continue to follow along the same old rut, or have the law-makers force on all mines rigid laws which are mere guesses at the real remedy. The first course will continue the present increasing toll in human lives, and the second is likely to do the same, besides forcing on the operators the practically useless expenditure of large amounts of money. This is economic waste, and is particularly unfair to the small operator.

The conditions in different localities vary. Legislation tends to take on a compulsory uniform form, and many mines are therefore forced to comply with regulations which are useless. The mine which is operated by shaft generally has to contend with conditions entirely different from those in the same seam where it is located above water-level and high in a mountain. One is likely to generate gas and become very dry, while the other rarely develops gas and is seldom dangerously dry. These are the two extremes; and as the seam goes under heavier and heavier cover and the area of solid coal increases, the conditions of shaft-mining will probably be more nearly approached. Certainly, when the truth is known, the operation of mines will be regulated in proportion as they are more or less dangerous. Some excellent laws already exist, but a problem is now confronted which defies solution. Possibly this very winter direful explosions may occur close together, and create such a popular storm of sympathetic but ignorant indignation that some very drastic though futile mine-laws may be enacted. To prevent such hasty action it is necessary to have recorded facts. The daily records here suggested would establish very closely for each particular locality the times of the year at which the pendulum of moisture swings to the side where the mines are robbed of moisture, and *vice versâ*. This knowledge could be applied to those mines in which it is evident that dry conditions are apt to reach the danger-point.

What temperature should be selected as perilous? In Table IV. it will be noted that the temperature of the mine-air varies from 50.5° in mid-winter to 60.2° F. in almost mid-summer. It is also to be noted that one reading of mine-air in January, 1908, showed a mine-temperature of 59° with an outside temperature of 41°. I will assume in this particular case that had the outside temperature been raised to 60° the inside temperature would have risen at least to the same point (60°). Now from Table IV. it is evident that with the same outside and inside temperatures at the mine under consideration, the ventilating-currents would rob the mine of moisture, since the percentage of saturation of outside air is uniformly less than the percentage of saturation of inside air. This assumption may not be justified when we have records of many readings, but the indications in its favor are strong. Furthermore, these humidity-

conditions may not be general throughout the Appalachian coal-field, but since they are the only ones for which data are available, let us assume, for the present purpose, that they represent average conditions. With this explanation, let us further assume that 60° outside temperature is that temperature which if reached will assure the robbing of the mines of moisture. This temperature will not cause a mine to become dry rapidly, but if a mine has been dry all winter and the temperature of 60° is reached in the spring, and if the temperature of any dust in a particular mine were raised to the ignition-point, an explosion would naturally result, the force or extent of which would be limited only to the dusty area in the mine; and if gas were present the violence would be increased. Had this condition of temperature prevailed in an excessively dry fall, in September or October, the same result would likely have been the case.

Assuming 60° as our critical temperature when a dry condition exists in the mines, let us consider Table III. This table shows that the total number of explosions in the last 10 years was 52, and of these 47 occurred either at 60° or below. This represents 90 per cent. of all explosions from all causes. Considering fatalities, 1,812 out of 2,126, or 85.23 per cent. of the total, were caused either at or below this temperature.

If we were able to eliminate all of those explosions which were due to gas, pure and simple, and not affected by dust, the figures would be far more conclusive. I have previously pointed out that I consider all of those explosions which occurred above the temperature of 60° F. as gas-explosions and not influenced by dust. The indication is that during the period under consideration we have not experienced a dust-explosion when the outside temperature was above 60° F. Close study of data (to be collected) may show that the temperature may fall from 5° to 10° below this point before danger of dust-explosions will be experienced.

TABLE V.—*Average Number of Fatalities for Certain Ranges in Temperature.*

Range in Temperature. Degrees Fahrenheit.	Average Number of Fatalities.
Up to 60°	38.6
Up to 50°	41.9
Up to 40°	46.06
Up to 32°	53.09

Table V. shows that as the temperature falls below 60° the average number of fatalities per explosion increases.

TABLE VI.—*Explosions at Various Ranges of Temperature.*

Range of Temperature. Degrees Fahrenheit.	Explosions. No.	Explosions. Per Cent.	Fatalities. No.	Fatalities. Per Cent.	Number Fatalities Per Explosion.
10 to 20	9	17.31	740	34.80	82.22
20 to 30	14	26.92	429	20.18	30.64
30 to 40	13	25.00	489	23.00	37.62
40 to 50	5	9.62	63	2.97	12.60
50 to 60	6	11.54	91	4.28	15.16
60 to 70	5	9.61	314	14.77	60.30
Total, 10° to 70°	52	100.00	2,126	100.00	40.90

Table VI. shows that as the temperature falls below 40° the number of fatalities increases greatly. It is my opinion that the increased number of fatalities listed between the temperatures of 50° and 60° is caused by the fact that there are included certain gas-explosions which swell the total. Even so, the data indicate that 60° of outside temperature is dangerous. The cause of this increase in number of fatalities as the temperature falls, naturally follows from the fact that mines become drier as the temperature falls, and the extent and violence of a particular explosion vary in some way directly as the dry area in the mine. The presence of gas, of course, will increase the violence. But back of all is the question of humidity, which is so closely linked with temperature in its rise and fall.

There must be a practical solution to this problem, and I believe it can be found. The method of spraying by means of a car, or by means of pipe-lines with sprays located at intervals, I regard as unsatisfactory and wasteful of water in the first case, and very expensive and cumbersome in the latter. The use of calcium chloride, $CaCl_2$, as a deliquescent is excellent for haulage-ways and partings, but it will prove very expensive if used in all working-places, in which most of the dust is generated and where the greatest danger from explosion lies. Whitewashing, and the use of slate- and shale-dust as a damper for explosive conditions, of course possess merit, but these are local preventive measures, and require too much time and supervision to be followed systematically.

I am inclined to the view that it will be necessary to install a blower-system such as is used in ventilating large office-

buildings, which heats the air to the required temperature. The air, having been heated to a temperature equal to the critical temperature, must be saturated with moisture at that point. This saturation would best be done near the intake, inside the mine. Long shallow pans might be tried, or a series of sprays arranged so that the current of air would be forced through the finely-divided water. The number of flat shallow pans or the number of sprays needed could readily be determined by experiment. This method of control would be less expensive, and more economical in the consumption of water, than the intricate watering-systems which are laid in each entry of the whole mine, and being concentrated in a small area it could be regulated automatically. It is not my intention to describe this system in detail here, but I may do so at a later time if the subject proves of sufficient interest.

The installation of such a system in dangerously dusty mines would reduce explosions from dust to a minimum. It may be found that certain portions of mines will still be persistently dusty, just as we find such spots, even in midsummer. These troublesome places can be dampened by a spraying-car or by some other local means of application. Moreover, in all cases and with whatever auxiliaries, that system of mining the coal should be practiced which will produce the minimum quantity of dust, and the method of firing and the explosives used should be determined by considerations of safety.

Conclusions.

1. There is a certain condition of the outside air, having a temperature and humidity such that when it is used in ventilating mines moisture will neither be deposited in the mines nor carried out of them. This condition I call the "critical condition." And this condition probably varies slightly according to latitude, altitude, and the seasons of the year. This range of variation will probably fall between temperatures of 50° and 60°.

2. As the temperature of the outside air falls below the critical point, mines will be robbed of moisture by the ventilating-current (and *vice versâ*).

3. While this lower outside temperature continues, the mine will become drier, and in proportion to the difference between

this lower outside temperature and the critical point, the rate of drying will be increased or diminished.

4. When the outside temperature is such as to make the ventilating-current dry the mine, this effect will be increased by increasing the volume of the current.

5. This critical condition should be determined, and a ventilating-and-heating system should be adopted which would raise the temperature of the intake air to the mine-temperature, and then saturate the air with moisture.

Dust-Explosions in Coal-Mines.

BY FRANKLIN BACHE,* FORT SMITH, ARK.

(Spokane Meeting, September, 1909.)

THERE seems to be in the public mind, and even in the minds of some coal-operators not experienced in mines subject to dust-explosions, a feeling that there has been something mysterious at the bottom of a number of recent American colliery-explosions. It has been declared in cases of accidents in mines regarded as particularly well equipped, in which every preventive precaution was said to have been taken, that the explosions were incomprehensible, and resulted from causes beyond the present knowledge of the practical coal-operator. But it is safe to say that no explosion has taken place which could not be explained by reasons well understood by most operators.

The U. S. Geological Survey Testing Laboratory, recently inaugurated at Pittsburg for the reported purpose of investigating scientifically the matter of explosions in mines, will discover no new explanation of explosions. It can, however, co-ordinate the previous investigations of similar boards in England and on the Continent; and it may do a vast amount of good by promulgating widely, and in a form comprehensible to the most unlettered miner, certain facts and obvious deductions that cannot but have an effect in reducing the number and extent of the mine-disasters which, in the last few years, have been so terrible in frequency and magnitude.

---

* President Bache-Denman Coal Co.

The only possible sources of explosions of any magnitude in coal-mines are: (1) explosives stored in quantity in the mine; (2) gases generated or liberated in the mine; and (3) coal-dust. Combinations of any of the three may, of course, take part in the result. The danger of any great loss of life from stored explosives alone is very remote. The simplest ordinary precaution forbids the presence, in one place underground, of any considerable quantity of explosives, and it would require an exceedingly large quantity to cause such an explosion as would kill any considerable number of men, distributed through the mine. (The effect that firing a quantity of explosives would have in stirring up and igniting the dust is another matter.)

Inflammable gas, and its explosive mixture with air, the old enemies of the miner, will doubtless always constitute a lurking danger; but in the years since Davy invented his lamp we have learned pretty completely the ways of fire-damp. With modern fans and the widely-diffused and almost exact knowledge of ventilation, fire-damp is not allowed to accumulate; and even in mines making gas so fast that explosive mixtures might be formed throughout the workings in a short time through any failure of ventilation, such failure is invariably guarded against by duplicate ventilating-plants. While the danger from gas is always present, we thoroughly understand it, and are taking all precautions that such a thorough knowledge suggests. The chance of gas-explosions affecting whole mines is very small. The danger to the individual miner, or to small numbers working in some section of a mine, is inherent in the work; and, do what we can with safety-lamps and well-directed air-currents, individuals here and there will be burnt. The effect of explosions of small quantities of gas in agitating and firing coal-dust, and the effect of gas in less than explosive proportions in the presence of a blown-out shot in a dusty mine, are again matters to be considered under the head of dust.

Coal-dust, therefore, alone remains as an inherent cause of great explosions. For many years this source of danger has been recognized, particularly in the deep mines of Great Britain; but it was not until the investigation of the Pocahontas explosion in 1884 that the subject aroused more than theoretical interest in the United States. Very little has been

done here in the way of scientific and laboratory inquiry into this matter, probably because the investigations on the other side had been thorough, and there did not seem to be much room left to better them. Recently the U. S. Geological Survey has undertaken, through a competent commission, to make an investigation which will concern itself greatly with coal-dust as an explosive agent. But practically we know already about coal-dust all that we need to know. What we want to find out is the best way of getting rid of the dust. We are quite aware that dust in a coal-mine is dangerous, and that the only way to make a mine safe in that respect is to eliminate the dust. We know that, next to eliminating the dust, the best thing is to undercut and wedge down the coal, using no explosive whatever, and that, if this be impracticable, the next best thing is to undercut the coal and use very little in the way of explosives, and that little as flameless and smokeless as it can be.

For reducing the danger from blown-out shots disturbing and firing the dust, we know only one means—namely, to undercut the coal, so that small charges of powder will suffice, and not "let the powder do the work." But it is difficult to see how we can get the coal undercut by the miner so long as he receives for a ton of coal, nearly all slack, the result of "shooting off the solid" a mighty blast of powder, the same pay as he receives for a ton of coal, nearly all lump, the result of undercutting the coal and using a small charge of powder.

So long as the miner gets as much per ton of "mine-run" coal with 80 per cent. of slack, which he has shot down from the coal-seam with the aid only of an excessive and dangerous amount of explosive, as he gets for a ton of "mine-run" coal containing 80 per cent. of lump, the result of first undercutting the coal and then letting the powder do only some of the work, he (being human) will "shoot off the solid," and when he finds that the enormous charge of powder is dangerous to his life he will endeavor to make coal-mining safe (to him) by insisting that the company shall procure men who do not mind the risk, to fire the shots. If the company does not see the matter in that light, the State legislature is induced to pass laws forbidding miners to fire their own shots, and compelling operators to employ other men to perform that dangerous duty.

If, in addition to procuring the passage of laws requiring the

employment of other men to fire shots which the miners are afraid to fire themselves, the latter would go a step further and get additional laws passed prohibiting the firing of any shots unless the coal had been undercut, as a protection for the lives of the shot-firers and the safety of the property, we would have few dust-explosions, and the shot-firer could no longer truthfully say, when told that his wage of $3 for two hours' work was pretty high, "The company don't pay me no wages. I just bet the company $3 a night that I won't get out of the mine. When the company loses, it pays me $3; when it wins, it buries me."

I have heard no rumor that any such combination of laws has been advocated by the miners before any State legislature. If such laws should ever be passed, it is needless to say that there will have to be an adjustment between the miners, the company, and the consumer as to how the cost of undercutting the coal shall be divided. One thing is certain: the undercut so much to be desired will not be made for nothing.

But pending the arrival of that semi-millennium we are not going to get rid of the miner's delectable sport of "shooting off the solid," with its frequent windy shots; and we shall continue to let the powder "do the work," which sometimes it surely does, as many a fatherless family can testify. Meanwhile, the miner has reduced his occupation from one requiring a considerable amount of skill, to one in which 90 per cent. of his labor consists of absolutely nothing but shoveling coal from the floor of the mine into a pit-car, a thing which could be done with a few hours' instruction by the most ignorant immigrant the day after he lands, for one-third of what is paid by the "scale" of wages to the average miner of to-day.

Since we do not seem likely to get rid of the blown-out shot, we must devote ourselves to getting rid of the dust. The natural first step, as I have shown, would be to make less dust in mining the coal by undercutting it and thus getting mine-run coal containing, say, only 30 per cent. of slack, as compared to "shooting off the solid" and getting 60 per cent. of slack; but being unable to arrange that matter with our miners, let us consider what to do with the dust we actually have, whether it be necessarily or unnecessarily made.

The remedy almost universally suggested and employed is

to water, or otherwise dampen, the dust. Laying pipe in the entries and watering the roadways thoroughly at frequent intervals is completely satisfactory if thoroughly carried out.

But the difficulty lies right there. The watering must be done at frequent intervals, and the pipe must be kept up to the face of the entries and kept in repair and renewed from time to time. This work becomes a matter of daily routine. The mine-foreman cannot give it his personal supervision at all times. To lay the dust thoroughly is a long and rather tedious job, as any one will testify who has seen how water will apparently run under the dust, run around it, float it—in fact, do almost anything but mix with it. Moreover, the most impalpable and most dangerous dust will be found lodged on the ribs and roof. In many mines, to wet the roof means to bring it down, with attendant damage and expense.

Scattering in the entries some hygroscopic substance, such as calcium chloride or common salt, has proved fairly efficient.

Moistening the air with steam or a spray of water before it enters the mine is not effective.

In the summer, when the outer air is warmer than the mine, the cooling of the air as it enters the mine quickly causes it to deposit, on the roof and elsewhere, near the intake, the surplus moisture which its lowered temperature no longer permits it to carry. Hence, a humid air-current cannot be taken through the whole mine. To add moisture to the air of the down-cast would be useless.

In the winter, no matter how fully saturated the air may be as it enters the mine, as soon as its temperature is raised to that of the workings it is no longer saturated, and, instead of depositing, it absorbs moisture. In winter, therefore, unless the temperature of the entering air be raised to that of the mine at the same time that it is saturated with moisture, it would be useless to moisten it.

It might pay to heat the air to the mine-temperature at the same time it is moistened (both could be done by steam-jets, and exhaust steam now wasted could be used), or the air could be moistened in the mine after it had traveled far enough to acquire the mine-temperature.

But both watering the entries and moistening the air are subject, in many localities, to the very serious objection that the

moistened roof will slack and fall, and become another source of danger and expense.

The best way to stop dust-explosions is to remove the dust, certainly from all parts of haulage-ways that are near enough to working-places to be affected by blown-out shots. It is not, in my opinion, necessary to remove the dust from rooms. The dust made in mining and loading the coal is comparatively coarse and safe. The dangerous material is the impalpable dust made by the continual grinding by passing men, mules, and cars on the haul-ways.

If we get the dust out of the haul-ways whenever it is in juxtaposition to possible blown-out shots, we shall do away with 90 per cent. of the danger from dust-explosions. If we allow no shots to be fired except when all the employees but the shot-firer are out of the mine, we shall greatly reduce the loss of life in case of an explosion. And if, in addition, we can prohibit the firing of any shots in coal not undercut, we shall have almost no dust-explosions at all.

Sweeping the roof, sides, and floor of an entry for a few hundred feet and loading the dust out every week or two is not an appalling task. And the providing of dust-proof pit-cars would not be impossible. We could even do away with the end-door, and dump the car even on self-dumping cages by making some changes in our dumping-arrangements. And some time, who knows, we may get the miner to see that it is his interest, as well as ours, to put a premium on mining more lump- and less slack-coal.

The suggestion has been made that we would have fewer dust-explosions if we reduced our ventilating-currents. This is doubtless true. The less ventilation the less drying-out of the dust and the less dust stirred up. But it has taken a century to get all hands connected with coal-mines thoroughly imbued with the desirability of ample air, so that we now have no great disasters resulting from gas. Let us not compromise and decide that, in order to save some lives from dust-explosions, we will sacrifice more to gas-explosions. The logical thing is to reduce in every possible way the making of dust, and remove from the mine that which is unavoidable.

I trust that the Geological Survey Commission, in emphasizing the danger of firing heavy charges of explosives, will make

its reasons so clear, express them so concisely and simply, and publish them in so many languages as to reach the understanding of every miner, even the most ignorant. At the same time, I trust it will avoid technical expressions, so that its conclusions and reasons may be readily understood by the non-mining public and by legislators. Under the pressure of public opinion, and through their own enlightenment, the miners themselves may consent to legislation limiting the charge of explosives and compelling the undercutting of coal, and may agree to receive less pay for the comparatively valueless slack they produce and more pay for the lump, on a basis just alike to them and to the operator.

Should the Geological Survey Commission accomplish this, or make any material step towards its accomplishment, it will have justified its existence a thousandfold. Incidentally, a great diminution in the millions of tons of slack now yearly made would do much to conserve the national resources, and by reducing mining-fatalities to conserve the greatest natural resource of all—the lives of workers.

Borax-Deposits of the United States.

BY CHARLES R. KEYES, DES MOINES, IOWA.

(Spokane Meeting, September, 1909.)

CONTENTS.

I. INTRODUCTION.

A COMPLETE transformation has taken place in the borax industry during the year 1908. A most remarkable factor in

this radical change in method of producing the crude borates has been its removal from the realm of industrial chemistry to the field of mining. With the development of extensive deposits of borate-minerals interstratified in thick sequences of Tertiary clays and sands, their winning becomes a strictly mining-enterprise, of the same kind and of the same certainty as the digging of coal or iron-ore.

The major supply of the world's production of commercial borax now may come from the United States. Although once all of the boric salts of commerce were laboriously extracted from the waters of saline lakes of the arid regions, or from the bottom-salts of desiccated ponds, it later was largely slowly leached from ancient desert shales and clays. During these periods the borax industry was a very hazardous and expensive vocation.

The discovery of large deposits of very pure, crystallized borate-minerals in the old Tertiary clays of southern California has enabled the main borax-supplies of the United States to be drawn from a single locality in the Mojave desert, near Daggett. The later finding of immense deposits of the crystallized mineral within easy access to good transportation-facilities promises not only to alter the character of the borax industry for the entire world, but to reduce the cost of production to less than one-third of the present figures.

Some of the extensive bedded deposits of borate-minerals recently discovered and investigated are located in a country that has been always one of the most inaccessible places of our domain. The geology of the region has been wholly unknown. Although borates had been recorded from the district, the importance of the deposits has never been determined. The geographic extent of even the shales carrying the borates, or likely to contain them, has only been suggested in the vaguest manner, and then with no association of commercial values. The stratigraphy of the borax-minerals is, therefore, at the present time, of exceptional interest. In the Death Valley district the Tertiary clay-beds, of great extent and thickness, are perhaps as finely displayed as anywhere else. Many of the geological facts associated with the occurrence of the borax are also worthy of more than passing notice.

The present account of the geology of the borax-deposits in

the United States originated incidentally in a commercial inquiry regarding the future of certain kinds of borax-supplies, undertaken for one of the chief borax-producing companies.

Later was begun a more personal inquiry of the more strictly geologic features of the occurrence of borate-minerals generally. Extensive investigation into metalliferous mining-properties in the neighborhood of certain of the larger borate-deposits gave opportunity to work out more in detail the geology of the districts in a way which before it was impossible satisfactorily to do. The results of these observations in the Death valley, the Mojave desert, and the Santa Clara valley are given herewith. The more strictly industrial aspects and the treatment of the borax-substances for the market will be described in a later paper, after the deposits of other parts of the world have been inspected, and especially those of South America, Turkey, Italy, Germany, and India.

II. Occurrence.

It is now nearly 50 years since borax was first produced in commercial quantities in the United States. Since that time, about 1864, the industry has undergone several distinct changes, and is now entering upon its fourth important stage.

During the early period, soon after the discovery of the presence of boric acid in the waters of Clear lake, in northern California, lake-waters were evaporated and the boric salts extracted from the residues. This method prevailed for the period from 1864 to 1872.

In 1874 it was found that the crusts formed on the surface of certain desert marshes were rich in boric contents. From 1872 to 1890 the chief boric-acid supply of this country was gathered from the bottoms of desiccated ponds in California and Nevada.

A score of years then passed before it was surmised that the marsh-deposits might be possibly naturally leached from the clay-formations which bordered the dry lakes. The clay-beds themselves then began to be exploited. The utilization of the boraciferous Tertiary clays continued from 1890 to 1905. Some boric acid is still obtained from this source.

In the newest period great deposits of very pure borate-minerals have been found imbedded, or interstratified, in old Ter-

tiary sediments; and large mining-operations have been already started which bid fair to control the borax industry of the world.

The Tertiary clays in which borate-minerals are found occur principally in southern California, but partly in Nevada. They extend in a rather broad belt, semicircular in shape, around the southern extremity of the Sierra Nevada and about 60 miles from that range. From Death valley and the Nevada boundary these clays are exposed at intervals through a distance of more than 300 miles, to the Pacific ocean at Santa Barbara, north of Los Angeles. Along the Furnace canyon on the east side of Death valley, in the Amargosa desert, in the low range of mountains north of Daggett, in the Mojave desert, in the Cajon pass of the San Bernardino mountains, and in the Santa Clara valley, which opens eastward from Santa Barbara, fine exposures give insight into the great areal extent of these deposits. For a decade or more Daggett has been the chief source of the borax-supply in the United States, but during the past year the Furnace Canyon and Santa Clara localities have been so extensively developed that the poorer deposits so long worked at Daggett are rapidly being abandoned. Moreover, better and more accessible deposits than any yet mentioned are ready to be developed.

The areal extent and relationships of the principal boraciferous beds are outlined in Fig. 1. In these localities the borate-layers are best exposed in the mountain-sides, where the stratified clay-beds, from 4,000 to 5,000 ft. thick; containing them have been tilted at a high angle and exposed to the erosive processes. Whether or not all of the borate-layers are in the same geologic horizon is not determined. Nor is it known with certainty that the several localities belong in the same geologic province. Presumably the clay-strata are continuous throughout the entire belt.

III. Geology of the Death Valley Borate-Region.

1. *Surface-Relief.*—The principal borate-deposits of the Death Valley region are interbedded with clay and friable sandstone formations of Tertiary age. These beds are best exposed in the Amargosa range, in Death valley, and in the Panamint mountains. This belt of mountain and valley is 125 miles long and

from 30 to 40 miles broad. To the east is the great Amargosa plain, and to the west the Panamint valley.

The district is a part of the Great Basin region, with its myriads of short, lofty mountain ranges, separated from one another by broad plain-like valleys. In this vast region, stretching from the Wasatch range on the east to the Sierra Nevada on the west, are found some of the most remarkable geographic features on the face of the globe.

The Sierra Nevada, rising from 10,000 to 14,000 ft. above mean-tide, constitutes the most conspicuous relief character of

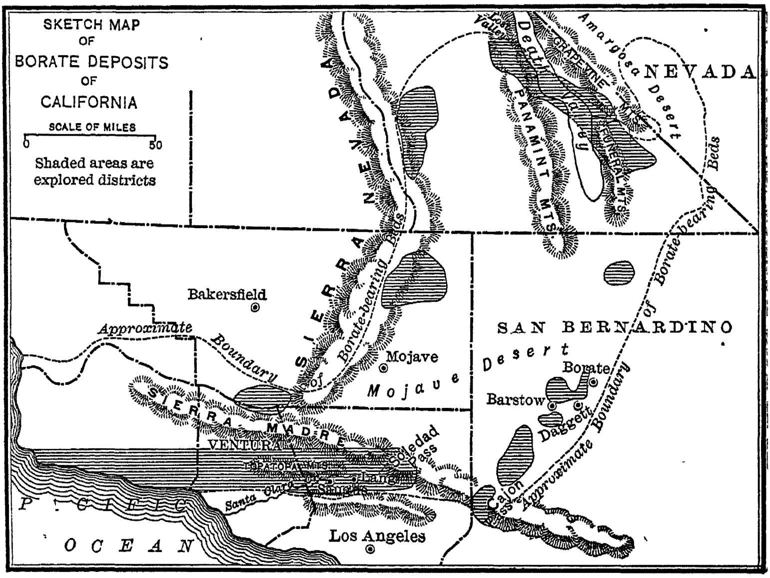


FIG. 1.—SKETCH-MAP OF CALIFORNIA BORATE-LOCALITIES.

the Western United States. Parallel to, and eastward of, this, majestic range there are other lesser ridges. The flat-bottomed valleys separating the abrupt mountains from one another are successively lower and lower as the distance increases from the main Sierra until Death valley is reached, which is the lowest depression of all, and the lowest point of any continent. East of the deep valley the intermont plains increase in elevation The general relief-contrasts thus produced are well shown in profile in Fig. 2.

The immediate area occupied by the soft boraciferous beds

is greatly diversified. The great Amargosa range, in which the principal deposits are found, presents to Death valley a very steep slope, which possibly represents a profound fault-scarp now much degraded. The north and south ends of the range are composed of hard clastics and eruptives separated by a belt of soft, infolded clays and friable sandstones, the belt of the latter passing diagonally across the mountain ridge. This slightly-resistant belt has been worn down from 3,000 to 4,000 ft. below the level of the crest of the range, dividing it into two somewhat distinct and nearly equal parts. The northern portion is usually known as the Grapevine mountains, while the southern part is called the Funeral mountains. The extremities of the Amargosa range fade out into the plains in the same manner

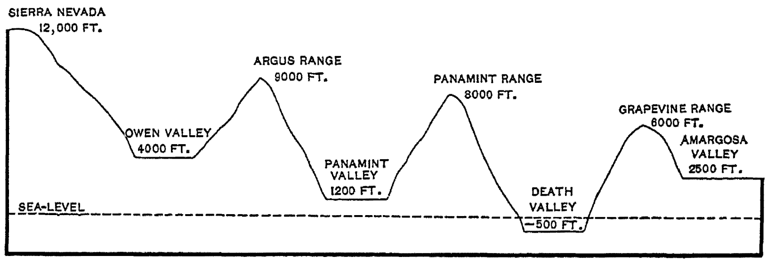


FIG. 2.—PROFILE OF THE DEATH VALLEY BORATE-REGION.

as do the neighboring ranges. It is in the soft middle belt that the chief borate-beds occur. (See Fig. 3.)

The valleys with which the boraciferous beds are particularly associated are the Death valley and the Amargosa valley. Like the majority of the intermont spaces of the Great Basin region and the Mexican plateau, these so-called valleys appear as vast plains seemingly as level as the sea. From the margins on all sides abruptly rise, without intervening foothills,[1] the lofty mountain ranges.

The most remarkable feature of the plains is the general absence of marked drainage-lines. Most of these basin-plains are true *bolsons*, such as are found farther southward in Mexico,[2] while some are *playas*, as the Spanish term them, having broad shallow sheets of water covering their central portions for a part of the year and at other times forming a bare mud-flat.

---

[1] *Bulletin of the Geological Society of America*, vol. xix., p. 573 (1907).

[2] *American Journal of Science*, Fourth Series, vol. xv., No. 87, pp. 207 to 210 (Mar., 1903).

Many of the intermont plains of the region have beveled rock floors,[3] and they are now believed to be formed chiefly through deflative erosion instead of tectonically, as they were long thought to be.

The drainage-ways leading into Death valley are many, short, and steep, and are occupied by water only briefly at rare intervals of heavy rain-fall in the neighboring mountains, or during the spring melting of the snows. In the mountains these drainage-ways lie in deep narrow canyons, but as they emerge into the great valley they are represented by shallow etchings on broad alluvial fans.

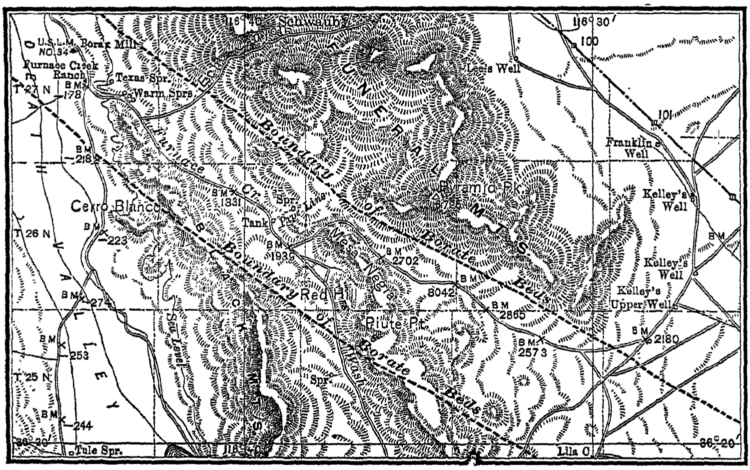


FIG. 3.—MAP OF FURNACE CANYON BORATE-DISTRICT.

Furnace creek has occupied the center of a narrow belt of clays and soft sandstones that traverses diagonally the Amargosa range. This canyon is relatively wide, and of much lower gradient than any of the other drainage-ways of the district. On each side of the main course are numerous deep ramifying canyons, cut in the thick clay-beds, the latter withstanding weathering in a remarkable manner. The crest of the canyon-walls on the east is capped by a thick basalt-flow, which forms a high cliff for a distance of several miles.

2. *Geologic Formations.*—The Death Valley region, which until quite recently has remained one of the least visited portions of

[3] *Bulletin of the Geological Society of America*, vol. **xix.**, p. 67 (1907).

the United States, has never received from travelers more than incidental mention. Aside from a few brief and scattered notes, nothing has yet been published regarding the formations containing the borate-minerals. Altogether the region still remains geologically a veritable *terra incognita.*

The geologic formations of Death valley may be grouped readily into five great classes. The first of these includes certain gneisses and schists, such as appear in the basal part of the Amargosa range north of Furnace canyon, and which are probably of Azoic age. The second group contains hard, and often somewhat metamorphosed, clastics of Palæozoic age; these constitute the foundation of the various mountain ranges of the region. The third class comprises extensive volcanic masses, mainly of Tertiary age, but some of very late extravasation; these are chiefly diorites, rhyolites, andesites, and basalts. To the fourth category belong great deposits of soft clays and sands, commonly regarded as of lacustrine origin, which attain a thickness of more than 4,000 ft., and which often have interbedded extensive basalt sheets. They are largely of Early- and Mid-Tertiary age. With the fifth group may be included all of the more recent clays, sands, and gravels which now mantle the plains and the valleys.

The geologic formations exposed in the vicinity of the Furnace canyon, where the chief borate-deposits are located, represent a total thickness of about 20,000 ft., systematically arranged as shown in Table I.

The fundamental complex, composed of the highly metamorphosed schists and gneisses, is but sparingly displayed in the immediate vicinity of Furnace canyon. The foundation of both the Amargosa and the Panamint ranges comprises mainly quartzites and hard blue limestones of Palæozoic age.

In the neighborhood of the Furnace Canyon pass, at the south end of the Grapevine mountains, the principal part of the mountain ridge appears to be made up principally of Cambrian rocks. A few miles to the north higher beds come in, including peculiar terranes, which seem to be the southward extensions of the Pogonip limestone of King[4] and the Eureka quartzite of Hague.[5]

[4] *Report of the Geological Exploration of the Fortieth Parallel,* vol. i., p. 232 (1878).
[5] *Third Annual Report, U. S. Geological Survey,* p. 254 (1881–82).

TABLE I.—*Geologic Formations About Furnace Canyon.*

	Age.		Thickness.	Rocks.
CENOZOIC.	Quarternary.	Late.	Feet. 500 200	Basaltic flows. Gravels, sands, and clays.
		Early.	1,500	Basaltic flows. Alluvial deposits. Basaltic flows.
	Tertiary.	Late.	1,000	*Playa* deposits.
		Mid.	2,500	Clays and sands with basalt-flows. Clays and marls.
		Early.	4,000	Andesites. Rhyolites. Diorites.
PALÆOZOIC.	Carboniferous.	Mid.	2,500	Limestones.
		Early.	300	Limestones.
	Devonian.	....	100	Limestones.
	Silurian.	....	100 400	Limestones. Quartzites.
	Ordovician.	....	3,000	Limestones.
	Cambrian.	....	2,500	Quartzites.
Az.	Huronian.	....	500	Gneisses and schists.

Besides the Ordovician rocks, there appear to be represented beds of Silurian, Devonian, and Carboniferous limestones. While the first mentioned are known to be entirely absent from the central portions of the Colorado plateau, and are generally regarded as not being present anywhere around its margins, recent inquiries show conclusively that beds of this age certainly occur at many points in the peripheral belt. In the Amargosa range possibly 400 ft. of limestone seems clearly referable to Silurian age. The exact relations of this section to the Lone Mountain limestone of the Eureka district are as yet undetermined, but it is most likely that the two are not coextensive.

At several places in the Amargosa range fossils have been discovered indicating the presence of Devonian beds. At least 100 ft. of strata is thus tentatively referred to this age. Devonian limestones and shales, long thought to be absent around the entire margin of the great dome of the Colorado plateau, have been found recently to be well represented.

Walcott,[6] for instance, has found rocks of this age in the Grand Canyon district, between the Cambrian Tonto formation and the Carboniferous Red Wall terrane. On the south side of the dome the Devonian beds are highly fossiliferous.[7]

The more recent formations of the region under consideration include three main groups of rocks: the early acid volcanics, the clays and friable sandstones and their associated deposits, and the interbedded basic lavas. The latter are to be clearly distinguished from the basaltic surface-flows of Quaternary age. The Mesozoic formations appear to be entirely absent, unless some of the rhyolites and andesites should finally prove to be partly of pre-Tertiary age.

In the immediate vicinity of Furnace canyon the early volcanic rocks find small exposure. To the northward, beyond Boundary canyon, they make up much of the Grapevine range. These rocks appear as numerous and successive flows of what is commonly termed porphyry. The complete sequence of these acidic lavas is at least 4,000 ft. thick, and consists partly of dull grayish and reddish andesites, but mainly of multicolored rhyolites. The first-mentioned flows are much the older, and may be eventually found to be Jurassic in age rather than Tertiary.

South and west of Furnace creek, in that part of the Amargosa range known as the Funeral mountains, the greater part of the mountains is composed of similar andesites and rhyolites, with considerable dioritic and monzonitic masses. The principal volcanics in the Panamint range on the west side of Death valley also appear to be light reddish monzonitic or granitic rocks.

All of these acidic volcanics seem to have been outpoured over more or less level plains, and the mountains partly elevated before the stratified clays and sands were deposited. The latter, of which the Furnace Canyon borate-bearing deposits may be regarded as typical, are widely distributed. They have a thickness, in this district, of probably more than 4,000 ft., and comprise mainly soft, yellowish to brownish sandstones, with numerous clay layers and greenish-yellow to whitish clays.

---

[6] *American Journal of Science*, Third Series, vol. xxvi., No. 156, p. 438 (Dec., 1883).

[7] *Idem*, Fourth Series, vol. xxi., No. 124, pp. 296 to 300 (Apr., 1906).

There are a few calcareous beds. The clayey portions of the deposits contain the borate-minerals. Interbedded with the sands and clays are many sheets of basalt, of which the individual layers often are 100 ft. thick.

The most recent terranes consist of, besides the wash from the mountains, some finer deposits of temporary lakes, and perhaps also *playa* deposits. These are separable into several distinct formations. Some borate-minerals are found in these beds, but thus far the deposits have proved to be unimportant. Extensive basalt-flows of a very late date cover large areas, and in many cases preserve the underlying soft clays from erosion.

3. *Geotectonics.*—Death valley is not only the lowest part of the Great basin, but the lowest area on the whole continent, being 500 ft. below mean sea-level. Contrary to prevailing opinion, the general tectonics of the Basin region is regarded as quite ancient. The so-called Basin Range type of mountain-structure is thought to be the exception rather than the rule, as an explanation of the rearing of the desert ranges. In the recent treatment of the origin of the Basin ranges the present relief-features were viewed from the stand-point of deflation, or wind-erosion, upon a planed surface composed of alternating belts of hard and soft rocks.[8]

The Death Valley district displays perhaps as well as any other area the distinctive characteristics of the so-called Basin Range structure. As it appears on first glance the geologic structure of the Amargosa range is that of a long, narrow, monoclinal block, tilted towards the east; that of the Panamint range, a huge mountain-block inclined westward; that of Death valley, a key-block 10 miles wide dropped down between, forming what the Germans call a *Graben* block. The general idea is represented in Fig. 4.

Militating against this explanation is the singular fact that no one has yet been able to point out any direct evidences of recent profound faulting along the sides of the *Graben*. This is a significant fact, also noted by Spurr,[9] as applying to the majority of the desert ranges assumed to represent typical so-called Basin Range structures.

---

[8] *Bulletin of the Geological Society of America*, vol. xix., pp. 63 to 92 (1907).

[9] *Idem*, vol. xii., pp. 217 to 272 (1900).

The marked flexings, faultings, and unconformities displayed in the region, it may be noted in passing, appear to have little direct influence upon the existing relief expression. It is only to the minor, later deformative effects that attention need be specially called in the present connection. While the strata of the region present notable folding, it is so overshadowed by

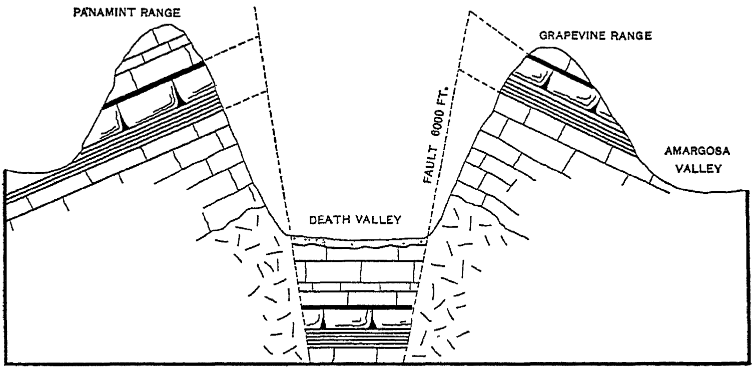


FIG. 4.—APPARENT GRABEN STRUCTURE OF DEATH VALLEY.

profound and frequent faulting that the phenomenon is not at first glance at all striking.

Some of the more gentle flexures may have been merely attendant or local phenomena of the major faulting. That there is some genetic relationship between the two is further suggested by the fact that the axes of the folds appear to be parallel to the trend of the adjacent mountain ranges. The Tertiary

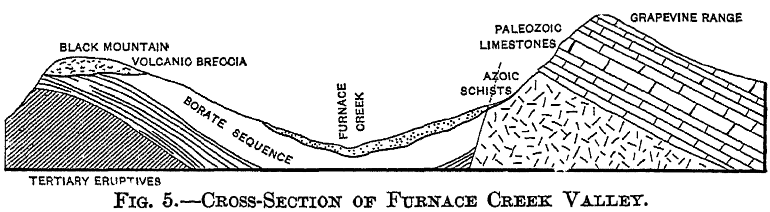


FIG. 5.—CROSS-SECTION OF FURNACE CREEK VALLEY.

clay-strata of the Furnace cañyon present the geologic cross-section shown in Fig. 5.

The older major faulting need not be described here, since only the minor faulting has a direct bearing upon the arrangement of the borate-beds. These dislocations have mainly an E–W. trend. They are so recent that in some cases they still impart to the surface-relief a characteristic feature. This is

particularly true when the surface is covered by basaltic flows, as, for instance, on the east side of Furnace canyon, and near the north end of the Funeral mountains.

Discordance in sedimentation is of special importance in considering the deposits of borate-minerals, for the reason that it has a direct bearing upon the economic exploration of the boraciferous field. In the section of more than 4,000 ft. of the Tertiary clays there are a number of great planes of unconformity, besides many minor ones. Some of the latter are so inconsequential as easily to escape notice. At the base of the clays-succession there are abundant evidences of profound unconformable relationships between these deposits and the indurated rocks beneath. Also, in the middle of the sequence, there is a very marked plane of like significance. At the top of the same section is a third great plane of discordance.

IV. Geology of the Borate-Beds of Furnace Canyon.

1. *General Features.*—The principal stratified beds carrying borate-deposits lie, as already stated, in a deep valley and canyon between two lofty mountain ranges. These boraciferous beds consist chiefly of soft clays and sands or friable sandstones, while the mountains on either side are composed largely of hard eruptives and metamorphosed clastics. Under conditions of normal humid climate, and in an elevated region, differential weathering alone would enable the weak formations to be eroded deeply within a very short time. In a dry country the same is also true, except that the erosive agent is chiefly wind-scour instead of water-action.

Furnace canyon, which traverses lengthwise the belt of clays trending diagonally across the Amargosa range, has now nearly bisected the great mountain ridge. This *arroyo*, or "dry-creek," has many lateral branches, deep and labyrinthine. The steep sides are produced partly by the undermining of the thick basalt sheet which still covers and protects from erosion many square miles of the soft clay-deposits. The vertical sections of the beds are many and are finely exposed. For the most part the local attitude of the layered deposits is readily made out.

Three rather distinct phases of the soft stratified beds occur. At the base of the section is a coarse conglomerate, which ap-

pears to be confined in its areal distribution chiefly to the eastern margin of the belt of finer deposits. It may form a part of the lower beds exposed in the Furnace canyon. It is possible, also, that it long antedates the Miocene terranes. For the present, until more critical evidence is obtained, it may be classed with the borate sequence. This conglomerate forms the south end of the Grapevine mountains, on the east side of the Furnace canyon south of Pyramid peak, where it has a thickness of more than 3,000 ft. It is also extensively exposed at the south end of the Funeral range, at the China ranch, where the Amargosa "river," for a distance of a dozen miles, cuts a deep canyon through the formation. At both of the

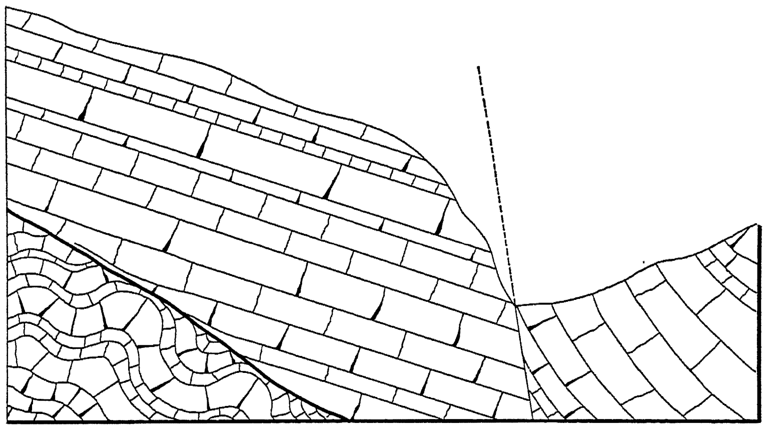


FIG. 6.—SHEARING AT SOUTH END OF GRAPEVINE RANGE.

localities mentioned the terrane is heavily bedded, strongly tilted SE., and nowhere markedly flexed.

The pebbles and boulders comprising the chief portion of the conglomerates are all water-worn and rounded, and are evidently derived mainly from the Palæozoic rocks. The general color is brownish or reddish. There are some fine-grained sandstone layers. The entire succession of beds is firmly cemented. The exact stratigraphic relationships of the conglomerate with the Palæozoics of the district are not as yet clearly understood. There are marked unconformable relations in some localities, but in one place at least, south of the Pyramid peak, there is a notable shearing, indicated in Fig. 6. Along the thrust-plane the conglomerate-beds contrast sharply with the contorted Ordovician limestones.

The second marked phase of the Tertiary stratified deposits has been called the older sand-series. The section is composed chiefly of yellow to reddish sandstones, rather massively bedded. These sandstones, so far as observed, nowhere merge into the underlying conglomerates; neither are there within them any conglomeratic facies. There are interstratified some minor clay layers. The upper surface of the sandstones is very uneven and is apparently sub-aërially eroded. The higher clays seem to lie upon them unconformably. The sandstones are well displayed in the Furnace canyon, about 15 miles above its mouth. The thickness of the beds probably exceeds 1,000 ft. Neither in the sandstones nor in the conglomerates beneath are there any indications of the presence of borate-minerals.

The third, and uppermost lithologic, phase consists of fine, alkaline, olive-green clays, which weather to pale yellow or white. Numerous olivine-basalt sheets, from 10 to 100 ft. thick, are interbedded. In the upper part of the sequence is much crystalline gypsum (selenite), thick beds of crystallized calcium borate (colemanite), and thin layers of limestone, probably of chemical origin. Thick beds of rock-salt are also reported to occur in several localities. So far as known, no fossils are found in any of the formations within the area under consideration.

2. *Geologic Structure.*—The strata in the Furnace canyon are all more or less disturbed. The tilting of the beds appears to be chiefly the result of late flexing. The main lines of dislocation, if such they be, blocking out the clay-deposits, strike nearly NW–SE. The main anticlinal flexure trends more nearly E–W. The axis runs nearly parallel to the north branch of the Furnace creek, or Black canyon, and 4 or 5 miles from it. A line drawn from the Lila C. mine to the Cerro Blanco, a distance of 25 miles, nearly coincides with this axis. It crosses the canyon obliquely, making a noteworthy feature of the local topography. The geologic cross-section, SW. from Pyramid peak, is represented in Fig. 7.

From the central core of the older sandstones, as displayed in the Furnace canyon, the younger succession of deposits containing the principal borate-beds dip on either side of the axis in opposite directions. On the east side of the canyon, basaltic lava-flows cover the soft clays, and rest upon their evenly-

beveled edges. The principal borate-beds have been cut through by the excavation of the canyon. Towards the crest of the Funeral mountains they are steeply upturned.

Near the summit of the older axis, where it passes beneath the basalt sheet at Piute point, the beds are so beveled that the

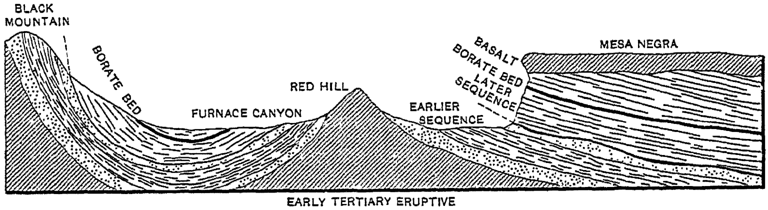


FIG. 7.—CROSS-SECTION OF FURNACE CANYON BORATE-DEPOSITS.

separated ends of the main borate-ledge on either side of the fold are 2 miles apart. The details are shown in Fig. 8.

Two miles to the north of the last-mentioned locality, under the Black mesa, where, near the top of the escarpment, the borate-beds again appear, the latter are sloping to the NE., as shown in Fig. 9.

In the Cerro Blanco, at the north end of the Funeral mountains, the succession of the borate-beds is finely displayed. An E-W. cross-section of the tilted strata is shown in the sub-

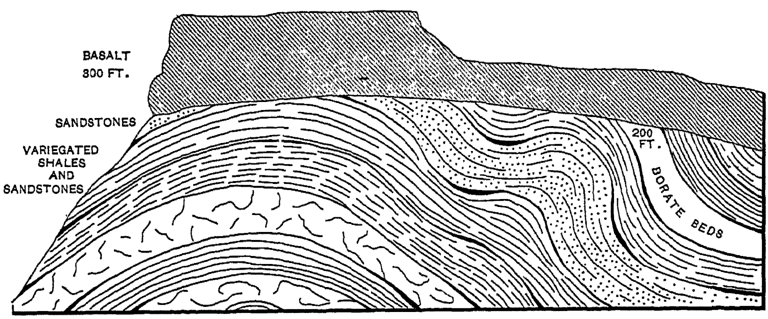


FIG. 8.—ANTICLINAL STRUCTURE AT PIUTE POINT. HEIGHT OF SECTION, 1,000 FEET.

joined diagram, Fig. 10. A notable feature of this sequence is the interbedded basalt-flows. The total thickness represented exceeds 3,000 ft. for the stratified clays alone. There are besides extensive deposits of old gravels, clays, and eruptives in thick sheets. The clay-deposits and sandstones present an alternation of soft and hard layers, which, with their present

attitudes, form a series of sharp parallel ridges separated by deep valleys. In this section also there appear at least two marked planes of unconformity.

3. *Ores.*—The richer borate-beds are from a few inches to 50 ft. thick. In the unweathered portions they consist of bluish clays thickly interspersed with milk-white layers, nodular bands, and nodules of crystallized colemanite, or calcium borate, which is termed locally "high-grade ore."

Through the strata carrying the coarse, crystallized colemanite, the clays are more or less highly impregnated with fine

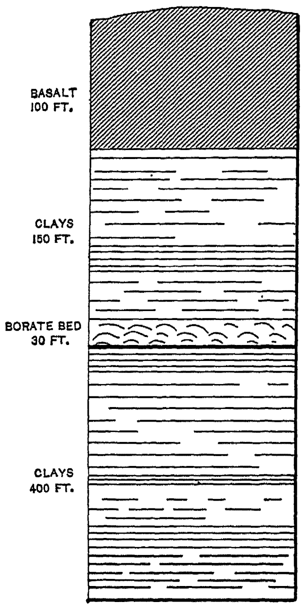


FIG. 9.—FACE OF BORATE-BEDS UNDER THE BLACK MESA.

particles of the borate-mineral, and yield, upon leaching, from 10 to 25 per cent. of anhydrous boric acid. This material is called by the miners "low-grade ore." While there are large quantities of this low-grade material in the Furnace Canyon district, it cannot be utilized to advantage in competition with the richer layers adjacent to it. At Daggett, however, similar clays carrying as low as from 6 to 12 per cent. of boric-acid content in a finely-divided form are exclusively mined on a large scale by two of the principal borax-producers.

Mingled with the coarse colemanite are often large amounts

of crystallized gypsum (selenite) in large plates. In places the gypsum becomes so abundant that the borate-minerals are all but completely obscured. Frequently, also, there are present large amounts of very pure lime, which sometimes forms compact bands resembling layers of ordinary limestone. There are to a greater or less extent associated in the borate-beds other alkaline salts. Special attention will be called to some of these salts in another place.

There appear to be in the Furnace canyon several distinct horizons at which the colemanite was deposited, but, so far as now known, there is only a single level at which the mineral was formed in large quantities. This horizon is exposed on

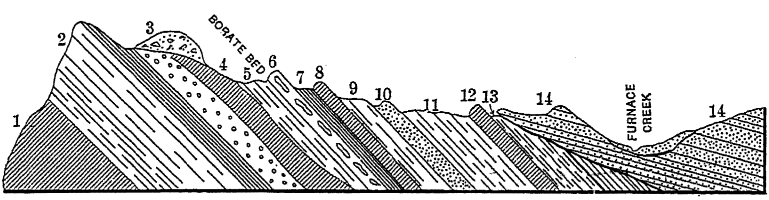


Fig. 10.—Tilted Borate-Beds at Cerro Blanco, Furnace Valley. Section 2 Miles Long.

both sides of the Furnace valley. Large quantities of the mineral have been already removed through the excavation of the canyon.

The clays associated with the borate-beds are all very fine and are entirely free from coarse material, the crystallizations excepted. These clays, when freshly exposed, are blue in color, but on weathering soon become olive-green, then yellowish, and finally nearly white. They seem to be of typical *playa* origin, very much the same kind as are being formed at the present day in many inclosed basins throughout the arid region.

4. *Typical Section.*—In their full thickness the later clay-deposits, or boraciferous beds, are nowhere exposed. The most complete section, showing the changes in the lithologic sequence, is displayed at Cerro Blanco, at the north end of the Funeral mountains. The relationships of the various beds are indicated in Fig. 10. The details of the stratigraphic succession are as follows:

Section of Borate-Beds at Cerro Blanco.

		Feet.
14.	Clays and gravels, pale reddish-brown and purple,	500
	Unconformity, very marked.	
13.	Clay, shaly, argillaceous, yellowish,	200
12.	Basalt, black, surface-flow,	30
11.	Clay, shaly, pale yellow,	500
10.	Sandstone, friable, red in color,	25
9.	Clay, shaly, yellow to green,	150
8.	Basalt, surface-flow,	100
7.	Clay, shaly, olive-green to yellow,	60
6.	Clay, shaly, colemanite in large nodules and nodular layers,	50
5.	Shale, argillaceous and sandy, buff,	300
	Unconformity.	
4.	Basalt,	200
3.	Gravels, coarse, little or no clay,	300
2.	Clay-shale, blue above, yellowish below,	1,000
1.	Andesite (exposed),	500

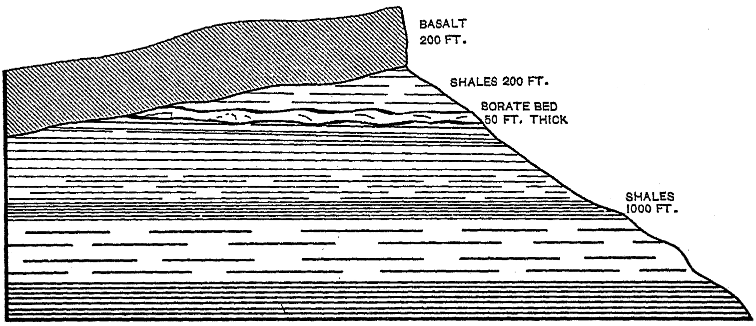


FIG. 11.—BORATE-BEDS UNDER MESA NEGRA, FURNACE CANYON.

On the opposite side of the Furnace valley, and 5 miles SE., under the Mesa Negra, the beds are inclined as shown in Fig. 11. While the thickness of the boraciferous bed is very clearly displayed, the subdivisions of the clay sequence are not so well shown as at the Cerro Blanco.

Two miles south of the last-mentioned locality, under the Black mesa, at a high sharp promontory called Piute point, the section is finely presented, as shown in Fig. 12.

At a mine-opening, 10 miles SE. of the Piute point, on the edge of the Amargosa plain, the clays are inclined about 20° E. In the sides of a low hill, where the borate-bed has been drifted upon, the richer colemanite-bearing stratum is 4 ft. thick, as illustrated in Fig. 13. Although the surface of the ground is obscured by the soil mantle, it is quite evident that

the workable stratum is thicker. The full section is well exposed near the mine-entrance.

V. Borate-Deposits of Lost Valley.

The great belt of yellow Tertiary clays traverses not only the Amargosa range but also the Death valley, and extends

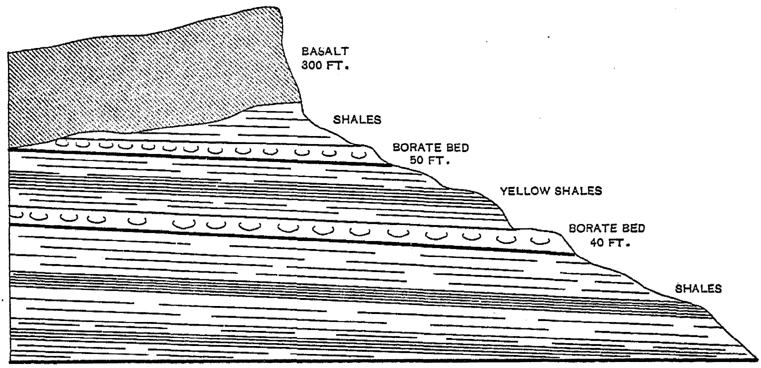


Fig. 12.—Details of Borate-Beds at Piute Point.

into the northern end of the Panamint range. On a spur of the latter, in an arm of Death valley called Lost valley, these clays are finely developed. At a point 25 miles from Cerro Blanco, in a NW. direction, borate-minerals occur. Thus far

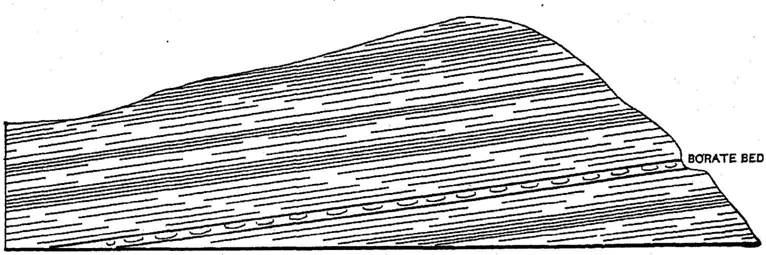


Fig. 13.—Drift on Inclined Borate-Bed at Lila C.

the deposits discovered consist mainly of low-grade material similar to the disseminated beds at Daggett. On account of the inaccessibility of the region at present little systematic exploratory work has been done.

VI. Borate-Deposits in Mojave Desert.

1. *Distribution.*—The peculiar yellow clays and sands, with which the borate-minerals are particularly associated, are widely

distributed in the Mojave Desert region. Only in the vicinity of Daggett, a station on the Atchison, Topeka & Santa Fé railroad, have the borate-minerals been carefully explored and opened up. It is this locality which has furnished, for a period of more than a dozen years, practically all of the borax obtained in the United States.

From the crest of the Sierra Madre, at the Cajon and Soledad passes north of San Bernardino, Cal., to the Furnace canyon, in Death valley, the yellow clays are exposed at frequent intervals in the low mountains which protrude above the broad expanse of the Mojave desert. Along the Mojave river, from the Cajon pass north to beyond Daggett, a distance of 75 miles, the outcrops of the formations in question are almost unbroken. They also occur on the opposite side of the desert-basin, on the flanks of the Sierra Nevada, 100 miles north of the Cajon pass. Since on the plains the yellow deposits are often found a few feet beneath the surface-mantle of wind-drifted soils, it is very probable that the same beds underlie the greater portion of the Mojave desert, especially the belt 100 miles wide extending from Daggett to Death valley and beyond.

The borate-bearing deposits are usually spoken of as lake-beds. Upon what grounds I do not know. Lithologically, they appear to be the same from Death valley to the Pacific ocean. Only in the western part of the Mojave plain have fossils been found, and these are marine Eocene and Miocene types. It seems probable that if strictly marine beds extend this far from the Pacific into the Mojave area, the Death Valley beds are also deposits of the sea rather than of extensive lakes in the process of desiccation.

The yellow clays of the Mojave region also contain inter-bedded basalt-flows similar to those occurring in Death valley.

2. *Geologic Structure.*—As admirably shown in the low mountains north of Daggett, the borate-beds are somewhat flexed and frequently infolded with the old volcanic sheets which once were surface lava-flows. The axes of the flexures are mainly E–W., and parallel to the trend of the great Sierra Madre line of uplift on the south.

At the mining-camp of Borate, 12 miles north of Daggett, the inclination of the strata varies from 15° to 50° southward. As shown in Fig. 14, the soft clays have not been deposited

around the foot of the mountains as recent lake-beds, but dip at a high angle directly into them near their summits. The crest of the range is formed by a thick sheet of eruptive rock, which constitutes a protecting cap for the weak clays beneath. The peculiarities of desert eolation permit differential erosion to go

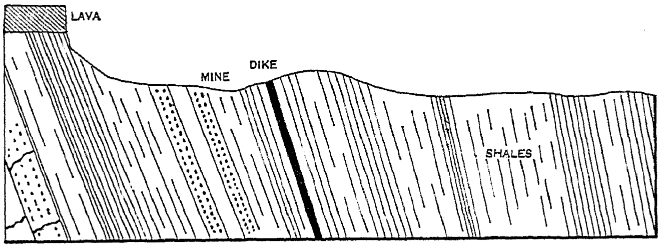


FIG. 14.—VERTICAL TERTIARY BORATE-BEDS NEAR DAGGETT, CAL.

on more rapidly in the arid country than in a normal moist climate.[10] It is estimated that under conditions of aridity erosion of soft rock-masses proceeds ten times as fast in a dry country as it does in a moist one; while under similar climatic circumstances the wasting away of hard rock-masses goes on only one-tenth as rapidly.

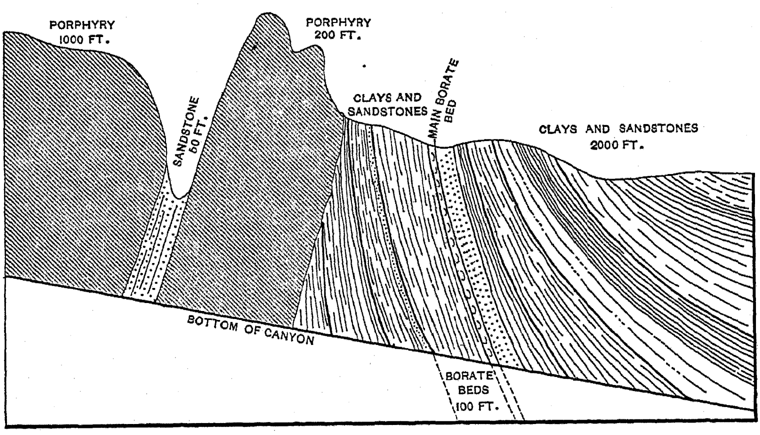


FIG. 15.—BORATE-DEPOSITS AT LANG, CAL.

The surface of the broad valley west of Borate and Calico and 10 miles NE. of Barstow, is over many square miles a true rock floor but thinly veneered by soil. The strata are highly tilted and evenly beveled. Wherever the more-indurated layers

[10] *Bulletin of the Geological Society of America*, vol. xix., pp. 63 to 92 (1907).

occur long ridges are found. At the mines of the American Borax Co. the section is finely displayed, as shown in Fig. 15. The beds dip 75° N. The yellow clays and sands are here more than a mile thick. There appear to be two well-defined borate-horizons separated by about 80 ft. of sandy shale; 15 ft. above the superior bed is a thin andesitic sheet scarcely 2 ft. thick. A short distance south of the mines is a high flat-topped hill. This is composed of the soft clays and sands standing on edge, evenly truncated and covered by a basalt sheet.

South of Daggett and Barstow, a distance of from 8 to 10 miles, the yellow clays and sands appear in force in several prominent E–W. ridges. These deposits have been prospected for borate-minerals, but have not as yet yielded workable bodies. The strata are only slightly inclined, seldom more than 5° or 10°. The soft yellow formations have interbedded numerous sheets of basaltic and andesitic lavas. In the tilted condition these faulted and resistant layers lying over weak deposits give rise to the long, sharp ridges. To the west these pronounced relief-features gradually melt away into the general surface of the immediate valley of the Mojave river, indicating that the lava-flows do not extend far in that direction.

3. *Ores.*—At the mines of the American Borax Co., NW. of Daggett, there are, as already noted (Fig. 14), in the yellow-clays section two distinct horizons from which the borate-materials are obtained. Both beds are about 5 ft. thick. The mineral is in a finely-divided state, the blue clay of the beds worked containing 10 or 12 per cent. of anhydrous boric acid. The 80 ft. of clays separating the two productive layers contain some borate-material, but not enough to make it profitable at the present time to remove. Near the old Calico gold-mine, 6 miles east, another borax-refinery is obtaining its crude material from similar deposits. There are doubtless other horizons in the general section which are borate-bearing.

At the mining-camp of Borate there are also two workable beds, about 50 ft. apart. Whether or not these two levels are the same as those worked at the American mine is not at the present time known. The high-grade borate here is mainly the calcium salt, occurring in nodular crystallized masses scattered through the blue clays. This nodular colemanite is now mined at depths of from 400 to 500 ft., and is treated at the refinery

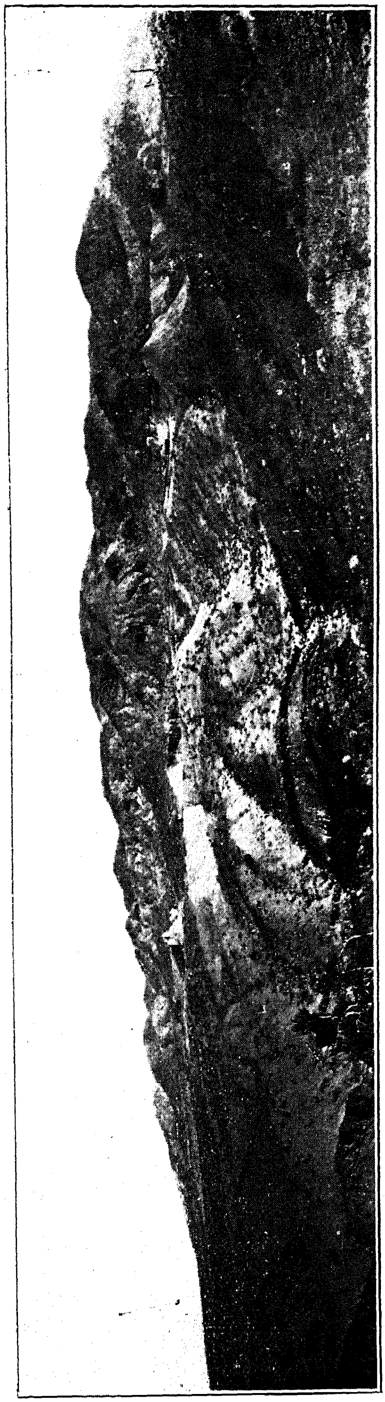


FIG. 16.—BORAX-MINES AT BORATE, CAL.

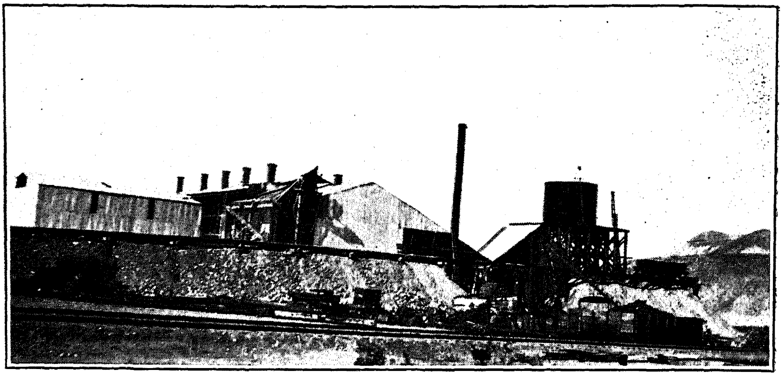


FIG. 17.—REFINERY OF THE AMERICAN BORAX CO., AT DAGGETT, CAL.

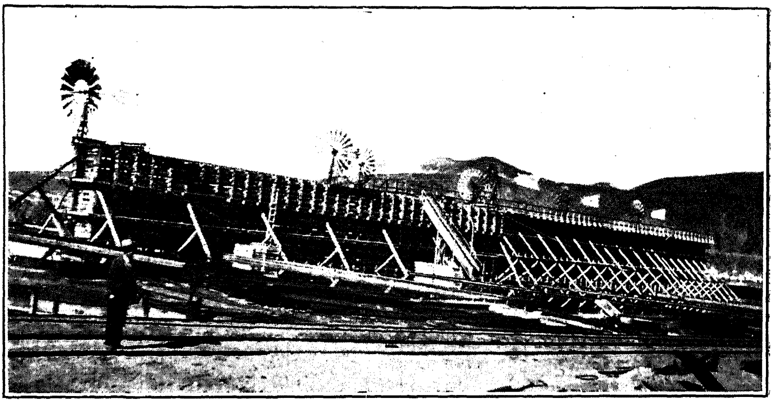


FIG. 18.—EVAPORATING-RACKS AT THE REFINERY OF THE AMERICAN BORAX CO., DAGGETT, CAL.

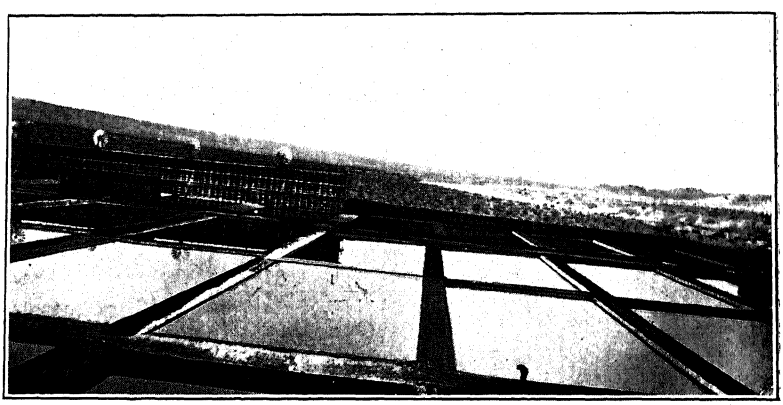


FIG. 19.—CONCENTRATING-VATS IN THE MOJAVE VALLEY.

near Daggett. The associated low-grade material out of which the crystallized colemanite is separated is not at present utilized, although elsewhere it is the low-grade boraciferous clays that are leached.

A photographic view of the borax-mines at Borate is given in Fig. 16; the refinery and the evaporation-racks of the American Borax Co., at Daggett, in Figs. 17 and 18, respectively; and Fig. 19 shows the general type of concentrating-vats used in the Mojave valley.

VII. Borate-Deposits of Santa Clara Valley.

1. *Distribution.*—Yellow sands and clays of Tertiary age are extensively involved in the foldings of the Sierra Madre extending from the Cajon pass to the Pacific ocean. The Santa Clara valley follows the southern foot of the Sierra, and eastward separates it from the western end of the San Gabriel range. On the north side of the valley the nearest range of the Sierra Madre is known as the Topatopa mountains. Near the eastern extremity of the latter, a few miles NE. of the junction of the two branches of the Southern Pacific railroad at Saugus, and 5 or 6 miles NW. of Lang station, important borate-deposits have been recently discovered.

The boraciferous formation is one of great thickness, variously estimated at different places at from 5,000 to 8,000 ft. It comprises mainly fine gravel-beds, more or less indurated, yellow sandstones, and yellow clays. These are traversed by intrusive masses. Judging from the outcrops at the south end of the railroad-tunnel under the Fernando pass, in the San Gabriel range, the beds immediately inclosing the borate-deposits near Lang appear to belong to the Vaqueros terrane, lately described by Eldridge.[11] This is of early Miocene age. Other parts of the Lang section may belong to the Pliocene Fernando formation of the same writer.[12] The geologic structure of the Santa Clara valley is complicated. The strata are profoundly disturbed, so that the detailed relationships of the formations in different parts of the region are not easily grasped without extensive investigations.

Compared with the yellow boraciferous clays of the Mojave desert and Death valley, the Santa Clara section contains much

[11] *Bulletin No.* 309, *U. S. Geological Survey*, p. 12 (1907). [12] *Ibid.*, p. 22.

more sandstone and conglomerate, which suggests that the borate-deposits of the latter district may be of somewhat later geologic age than the former.

2. *Geologic Structure.*—Notwithstanding the recency of formation and the great thickness of the yellow clay and sands in the Santa Clara district, the strata have been severely flexed and profoundly faulted. Several great unconformities tend also to vastly increase the complexity of the stratigraphy. The general tectonics of the Topatopa range, a few miles west of the borate-producing locality, is well displayed in the cross-section near Piru, modified from Eldridge, Fig. 20.

A short distance north of the Lang borate-belt volcanic rocks abruptly take the place of the soft clays and friable sandstones.

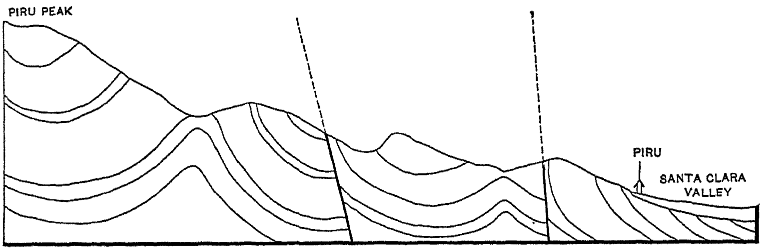


FIG. 20.—GENERAL STRUCTURE OF THE SANTA CLARA VALLEY, CAL.

In these porphyries considerable metal-mining is carried on. The strike of the strata is nearly E–W., but at the borate-mines there is repeated off-setting of the borate-stratum by faulting, having a trend NE–SW., and of several hundreds of feet lateral displacement in each case.

3. *Ores.*—At the point at which the borate-mines are opened the rocks are very much disturbed. The strata are abruptly upturned against a great basic dike, Fig. 15. As observed along the base of the canyon cutting the deposits transversely, the succession of layers is as follows:

Borate-Section at Lang.

	Feet.
6. Sandstone, hard, drab, ...	25
5. Clays, yellowish, with thin limestone-lenses, and several arenaceous layers, ...	100
4. Clay, bluish, carrying abundant colemanite nodules and layers, ...	4
3. Clays and sandstones, yellowish, often calcareous, . . .	200
2. Porphyry dike, ...	200
1. Sandstone, indurated, brown (exposed), ...	50

Mining is now also carried on by means of shafts, as in the cases of the Daggett and Borate localities. In a deep narrow canyon which cuts across the almost vertically-disposed boraciferous bed, and near the bottom, tunnels are driven into the canyon-walls on either side. The mineral is then stoped down, just as the larger ore-bodies are removed in metal-mining. The exceptionally good transportation-facilities, and the high-grade of the material, well combine to make this locality, for many years to come, the principal source of borax in the United States.

VIII. Borate-Deposits in Ventura County, Cal.

On the north side of the Sierra Madre, in the extreme corner of Ventura county, Cal., there occur extensive beds of the yellow clays and sands, similar in all respects to those found on the south slope of the same mountain system and in the Mojave desert. The strata are all highly inclined, and the colemanite is scattered through the blue clay in small nodular crystallizations. The locality has been worked for a number of years, and has furnished a considerable amount of material for the manufacture of boric acid. Since this point is 75 miles from the railroad, the operation of the mines has been conducted under great difficulties, but these in large measure are soon to be removed.

IX. General Geologic Occurrence of Borax.

1. *Original Sources.*—As a title in commerce the name borax is usually applied not only to the substance borax, $Na_2B_4O_7$, itself, but to the several boron compounds which are capable of being readily converted into the sodium tetraborate.

The element boron is widely distributed through the earth's crust. Commonly, however, it is found in such small quantities as to be almost inappreciable. As all of the boron salts, from which the article of commerce is derived, are quite soluble under ordinary climatic conditions, no valuable deposits of these salts occur in normally moist lands. As geological deposits the salts of boron are possible only under climatic conditions of extreme aridity.

Sea-water is known to contain minute amounts of borax. According to Forchhammer,[13] boron is one of the 27 elements the presence of which he detected in the waters of the ocean.

[13] *Philosophical Transactions of the Royal Society*, vol. clv., p. 208 (1865).

Certain of the rock-forming minerals have boron as an essential constituent. Its vapors are regarded as highly important mineralizers in the metamorphism of rocks and in connection with the formation of many ore-deposits.

Boric acid is a common exhalation accompanying volcanic eruptions. Vulcano, Stromboli, Etna, Vesuvius, and other active volcanoes in different parts of the globe give it forth in notable amount.

Many warm and brine springs give forth waters carrying in solution very appreciable quantities. The waters from the great Ash Meadows springs, in SW. Nevada, contain so much borax as to be noticeable to the touch. Roth,[14] especially, has called particular attention to the presence of borates in some of the brine-springs of Germany.

In the drier regions of the globe many of the bitter-lake waters contain considerable amounts of borax. This has been concentrated through long-continued evaporation. In some of these shallow lakes the borax forms in well-defined but scattered crystals in the muds of the bottom. In the old Tertiary clays of the West borates appear to have originated in large deposits in this way.

Geologically, deposits of borax derived from four of the five original sources just enumerated are unimportant. While formerly most of the borax of commerce was obtained from solfataric vapors and from the evaporation of strongly saline waters, little from these sources is now collected. At the present time the greater part of the world's supply of borax comes from the arid regions, where alkaline lakes in the last stages of desiccation yield either borax direct or borates from which it may be artificially derived.

2. *Solfataric Borax.*—Boric acid occurs as a sublimate in lava-cavities and cracks around active volcanoes. Acidic magmas in cooling give off such appreciable amounts of boric vapors that these, together with those of fluorine, chlorine, etc., become important "mineralizers" of the rocks through which they pass. So early as 1846, Élie de Beaumont, the famous French geologist, emphasized the activity which must be displayed by such vapors as those of boron, phosphorus, and fluorine in being

[14] *Allegemeine und chemische Geologie*, vol. i., p. 442 *et seq.* (1879).

expelled from consolidating granite magmas.[15] Since that time others have expressed similar views.

It is now a well-established fact that the borate-producing localities of the world are also districts in which volcanic activity has not yet ceased. The extent to which boron compounds occur at these places may be judged from the statement that the vapors in some situations are collected in commercial quantities, as in the Maremma of Tuscany. The vapors, as they issue from the *saffioni*, are passed through vats of water, which eventually become charged to the extent of 2 per cent. with boric acid, when the waters are drawn off and the process repeated.

3. *Lacustrine Borax.*—Lake-waters containing small percentages of boric acid have yielded some of the principal borax-supplies. In the United States the most noteworthy of such occurrences are at Clear lake, in northern California, and at Ragtown lake and Sand springs, in Churchill county, Nev.

At the Clear Lake locality the crystals of boric acid occur abundantly in the muds of the bottom of the lake. These muds are pumped out, washed, and sent to the refining-plant. The waters are also boiled in small vats and the boric acid finally crystallized. In Nevada the lake-waters were pumped out upon a plain and allowed to evaporate in the dry air.

4. *Marsh Borax.*—From the *playas* of the arid regions the major part of the borax-supply was formerly obtained. The bottoms of desiccated lakes are often, for a part of the year, covered by a few inches of water. The alkaline crusts which gather upon the floors upon complete evaporation of the waters are harvested and sent to the refinery. The material thus obtained is usually the native borax, $Na_2B_4O_7$, mixed with a number of other salines.

Half a century ago, when the principal portion of the world's supply of borax came from Thibet, in central Asia, it was from such lake-floors that the unrefined material was chiefly gathered. In the United States the main supply was for many years obtained in a similar way. Searle marsh, in the NW. corner of San Bernardino county, Cal., has long been noted for this class of borax.

15 *Bulletin de la Sociét' géologique de France*, Second Series, vol. iv., p. 1249 *et seq.* (1846–47).

It is generally assumed that such salinas as Searle marsh are the final remnants of former extensive lakes. According to the latest observations and deductions concerning the evolution of desert relief-features, it seems more probable that the majority of such salinas are due directly to the fact that eolian erosion has encountered ground-water level, permitting their constant evaporation just at the surface of the ground without forming open bodies of water.[16]

5. *Terranal Borax.*—Borates forming old geological deposits are now known to occur in a number of places in the arid regions. The layers, imbedded with shales and sandstones, are associated with gypsum, rock-salt, and other salines deposited from desiccating bodies of water. The bedded borates of California are the most important deposits of the kind known. They form geologic terranes in the strictest sense of the word. It is from this source that the world's supply in the future may be expected mainly to come. The laborious harvesting of lake-muds and thin surface-crusts will soon be a thing of the past. Borax-gathering now becomes a strictly mining industry.

As more fully stated in another place, large bodies of water are known to have existed in very recent geologic times in many parts of what are now eastern California and western Nevada. The smaller of these inland seas, for some of them were cut off from the ocean, soon became bitter-lakes, and finally dried up altogether.

As such bodies of water pass from the stage of saline lakes to that of complete desiccation many interesting precipitations take place. According to the most recent investigations on the subject, ordinary gypsum begins to be deposited on the lake-floor when about 37 per cent. of the water has evaporated. Then as the water progressively reaches the point of saturation for other salts they are thrown down in turn. Finally, when 93 per cent. of the water has passed off, common salt is deposited.

The most frequent succession of salts thrown down by progressive evaporation of saline waters of inland seas is: 1, boracite; 2, anhydrite; 3, gypsum; 4, sylvite; 5, halite, 6, kieserite; 7, polyhalite; 8, kainite; 9, carnallite; 10, tachyhydrite.

Contrary to long-accepted opinion, the various salts in saline

[16] *Bulletin of the Geological Society of America*, vol. xix., p. 90 (1907).

waters under conditions of arid climate are not precipitated in inverse order of their solubilities. The relative amount of the several elements in solution has a prime influence. This differs widely in different basins, so that under the same climatic conditions the same succession of salts does not always appear. Time is a noteworthy determining element. Temperature also plays an important rôle; and pressure has some influence.

The notable factor to be taken into account in considering the general sequence of the salts thrown down in bitter-lakes is the early appearance of the borates.

X. Chemistry of Natural Borates.

1. *General Considerations.*—Since the borates which supply commerce with most of the raw materials for conversion into borax as it is used in the arts now come from old lake-beds or inland-sea deposits, their chemical relations and development are quite like those of saline deposits generally. While a general sequence of salts in the precipitations from complex saline waters has been commonly regarded as established, it is now known that this succession is not everywhere invariably the same. Neither is the sequence in inverse order of solubility, as it was long thought to be.

The experiments on evaporating large quantities of sea-water carried on many years ago by the celebrated Italian scientist, Usiglio,[17] are well known. The results obtained by this chemist have been widely accepted; but more recent tests prove that they are not of so wide application as was at first supposed. Careful comparisons show that the artificial processes do not correspond exactly to the natural ones. This fact recently led the German chemists, Van't Hoff, Meyerhoffer, Hindrichsen, and Weigat,[18] to conduct exhaustive researches on the salt-formations in nature. Very interesting results were obtained, which throw a flood of light upon the subject, and offer satisfactory explanations to many hitherto little understood phenomena.

Among the important factors which Usiglio, and others who have been especially interested in similar experimentation, did not take into consideration were: 1, the composition of the

[17] *Annales de Chimie et de Physique*, Third Series, vol. xxvii., pp. 92 to 107 (1849).

[18] *Sitzungsbericht der königlich preussischen Akademie der Wissenschaften*, 1897.

saline waters; 2, the solubility of the compounds present; 3, the time allowed for concentration; 4, the temperature at which saturation for a given salt took place; and 5, pressure under which crystallization began. Since the recent chemical results have such a direct bearing upon the saline deposits under consideration, they may be briefly summed up here.

In the great salt-deposits of Stassfurt, Germany, which were chiefly investigated, it was found that in the succession of strata four very distinct zones were recognizable. These, beginning at the bottom and named after the principal salt found in them, were the anhydrite zone, the polyhalite zone, the kieserite zone, and the carnallite zone. In all of these zones rock-salt is found. There are also other salts present which are regarded as of secondary formation.

The desiccated inland-sea deposits of the Great Basin region of western America have not been as yet investigated in detail to determine the full variety of salts and their relationships. However, sufficient is known in the case of the borate-deposits of the Death Valley district to state something regarding the peculiar conditions existing at the time at which the salts belonging to the first or lowest zone were precipitated. This zone is the one containing, besides anhydrite, the borates, gypsum, calcite, and some other salts in which lime is an important constituent.

2. *Composition of Saline Waters.*—Were it merely oceanic waters with which we had to deal the chemistry of natural salines would be very simple. By not taking into account the calcium salts the composition would be identical the world over. The composition of the waters of bitter-lakes is very much more complex and varied. Many new conditions are introduced. Inclosed bodies of water, especially those of the very dry regions of the earth, receive compounds in solution from the surrounding elevations that vary greatly in every case, and according to the composition of the rocks, or geologic terranes. In every known instance some one salt greatly predominates.

Instead of the various salts being precipitated in inverse order of solubility, it appears that in a given solution the component which is greatly in excess is the one that is most likely to reach the point of saturation first, and hence will be the first to crystallize out. As Van't Hoff has recently clearly shown,

concentration will continue until the water reaches the point of saturation for a second salt, when that also will commence to be precipitated. If for the moment we can neglect the other salts, in order to give the problem its simplest form, it is from this point onward that the water remains with the composition unchanged. The water gradually evaporates and the salts continue to fall until complete desiccation has taken place.

3. *Solubility of Components.*—There is a wide-spread opinion among scientists that the salts which crystallize out of saline waters in the arid regions of the globe are merely in solution, and that merely the proper point of concentration is required to be reached in order to precipitate a given salt. Such, it has been already intimated, is not really the case.

Recent observation has conclusively shown that in the desiccation of some saline waters certain salts which naturally would be expected to be found do not appear at all. In other cases compounds entirely unexpected are actually deposited. Under one set of physical conditions the waters of bitter-lakes as they evaporate may throw down a certain series of salts, while under slightly different conditions the same saline waters may deposit an entirely distinct series of compounds.

The first-mentioned results are rather unduly emphasized on account of their being the outcome of laboratory-experimentation also. Here the physical conditions are always very nearly uniform, and the methods of chemical procedure fixed. In nature there is no such uniformity of conditions as is found in the laboratory. In consequence there are many departures from the artificially-conducted tests. Solubility is also a function of temperature, and varies in degree very greatly, as all laboratory-work shows.

4. *Time-Element in Water-Concentrations.*—In nature the time-factor in the determination of precipitates in solution is probably very much more important than is commonly assumed. In the chemical laboratory time is of necessity practically eliminated in all experimentation, and as a consequence very erroneous conclusions are often drawn regarding the chemical processes at work in the earth's crust and the results attained.

The unexpected chemical reactions in nature are as noteworthy in the desiccation of saline waters as they are among the rock-magmas in the process of solidification. Among the

last mentioned granite alone may be cited out of the many known examples. It is shown that an acidic magma, owing to the presence of aqueous vapor, the high pressures under which alone granite can form, and the long time that must manifestly pass, may cool down considerably below the temperature required to crystallize out certain minerals under ordinary dry-fusion conditions. Thus quartz, which should be formed quite early in the normal sequence, can be the last to crystallize, solidifying the whole mass into solid rock. This principle was long ago formulated by Scheerer,[19] who later advocated it at greater length and in greater detail.[20] It was subsequently confirmed experimentally by Èlie de Beaumont, Daubrèe and others, as well as by some more recent investigators.

In the case of similar retardations in crystallization of salts in saline waters under much simpler conditions than those existing among molten materials, recent inquiry has clearly indicated that such phenomena occur very much more frequently than was ever surmised. Length of time, however, is not the only determining factor in these cases.

In the formation of natural salts in desiccating lake-waters the time-factor must be regarded as of prime importance. To it must be ascribed the presence in the sequence of saline deposits of certain salts which never appear in the laboratory-trials. Concerning the saline deposits of the old inland seas of the Great Basin region, this time-factor explains much that previously was very obscure.

5. *Effect of Temperature.*—The general influence of temperature in effecting the crystallizations in saline solutions need not be dwelt upon at length here. Effects of the high temperatures are now well known. Effects of slight changes of a few degrees, within the limits of the ordinary temperatures as they are known in saline waters of the arid regions, have not been so well understood or considered.

At normal temperatures saline waters of the desert basins may deposit a certain number of salts and in a certain sequence. Under conditions of 20° or 30° increase waters of identical composition in the process of desiccation may give rise to some

[19] *Poggendorff's Annalen der Physik und Chemie*, vol. lvi., pp. 479 to 505 (1842).

[20] *Bulletin de la Société géologique de France*, Second Series, vol. iv., p. 468 *et seq.* (1846-47).

entirely new minerals. At the same time, at the higher temperature, some of the salts which commonly appear at lower degrees of heat do not form at all. Within certain limits the salts derived from evaporation of the waters of saline or bitter-lakes may be regarded as indices of the temperatures of the waters at the time the deposits took place. Hence, it is possible to use deposits of this kind as factors in the determination of geologic climate.

Temperature of saline waters has also a very important bearing upon the paragenesis of many of the minerals which are commonly associated in old lake-beds or deposits of inland seas. The gathering of winter and summer sodas in some of the alkaline ponds of Wyoming and elsewhere forms a good illustration. Just what part temperature has played in the formation of the borate-deposits of California has not yet been definitely determined, but it is thought to be highly influential.

6. *Effect of Pressure.*—The effect of pressure in the formation of saline materials in saline lake-waters can hardly be so great as it is in the cases of many other geologic deposits. Variations in pressure must be quite negligible, because the bodies of water of this kind are comparatively shallow when the salts begin to form. In laboratory-experimentation pressure is usually eliminated altogether.

XI. Occurrence of Other Commercial Salines.

The Tertiary boraciferous formations of southern California are the most remarkable and most extensive in the world. They were formed under conditions of an arid climate in a great shallow arm of the Pacific ocean that had been cut off by the upheaval of the mountain ranges along the coast. The inland sea was long in drying up, and perhaps had frequent connection with the ocean, as is shown by the enormous thickness of the terranes carrying the borates. The disappearance of the water may have been more rapid than the thickness of the deposits suggests at first thought, for the reason that as an accompaniment of the evaporation of the waters in an excessively dry climate there may have been a filling-up of the basin by the prodigious quantities of wind-borne dust derived from the neighboring deserts. It is not to be inferred that, since the

Tertiary clays and sands are between 5,000 and 8,000 ft. thick, the waters in the beginning were at least of the same depth, but rather that the arm of the ocean and afterwards the inland sea was always very shallow, and that as the area was filling up the waters continued to rest on the surface, rising with the rise of the bottom. This postulates a gradual sinking of the foundations of the region, and the truth of this is indicated by the general tectonics of the region.

The entire field of the Tertiary clays in southern California is capable of great results from systematic prospecting and exploration for commercial salines other than calcium borate. The inferences to be drawn from the modern conceptions of the deposition of salines are that with proper inquiry a large series of natural salts may be discovered. The calcium borate-beds are easily passed over unnoticed unless special care be taken to look for them. Other borates are found even more valuable than the colemanite. Extensive rock-salt deposits are already known, as are those of purest gypsum, anhydrite, and calcite. In some of the bitter-lakes immense bodies of soda and magnesia of the kind known mineralogically as blœdite, are among the most wonderful deposits recently found. In one small lakelet scarcely a mile across it is estimated that more than 1,000,000 tons of this mineral is readily available.

The Geology, Mining, and Preparation of Barite in Washington County, Missouri.*

BY A. A. STEEL,† FAYETTEVILLE, ARK.

(Spokane Meeting, September, 1909.)

DURING the summer of 1905 I was employed by the U. S. Geological Survey to investigate the geology, mining, and preparation of barite in most of the fields of the United States. The Eastern districts have been more or less completely described; the Virginia field in detail by Thomas L. Watson, in his paper, Geology of the Virginia Barite-Deposits,[1] and other fields in abstract, including statistics of production and references to current literature, by various authors.[2] There have also been published a few details of the occurrence of barite in the lead-mines of Washington county, Mo.[3] The other accounts of this district are erroneous and meager, and since this field is the most important one in the United States at the present time, a detailed description of the local geology seems doubly desirable.

Barite-deposits are scattered over nearly the whole of Washington and adjoining counties, but the product is all mined from the areas indicated upon the sketch-map, Fig. 1, and is shipped from the stations of the St. Louis, Iron Mountain & Southern railway, between Blackwell and Potosi, from 45 to 60 miles SW. of St. Louis.

The topography is for the most part gently rolling, with slightly graded streams, usually less than 200 ft. below the higher hill-tops. Along Mineral fork and Indian creek are some steep bluffs and cliffs, affording good exposures of the

---

* Published by permission of the Director of the United States Geological Survey.

† Professor of Mining, University of Arkansas.

[1] *Trans.*, xxxviii., 710 to 733 (1908).

[2] *Mineral Industry*, vols. ii., viii., x., xiii., xv., and xvi., under the caption "Barytes."

[3] *Report of the Missouri Geological Survey*, vol. vii. (1894).

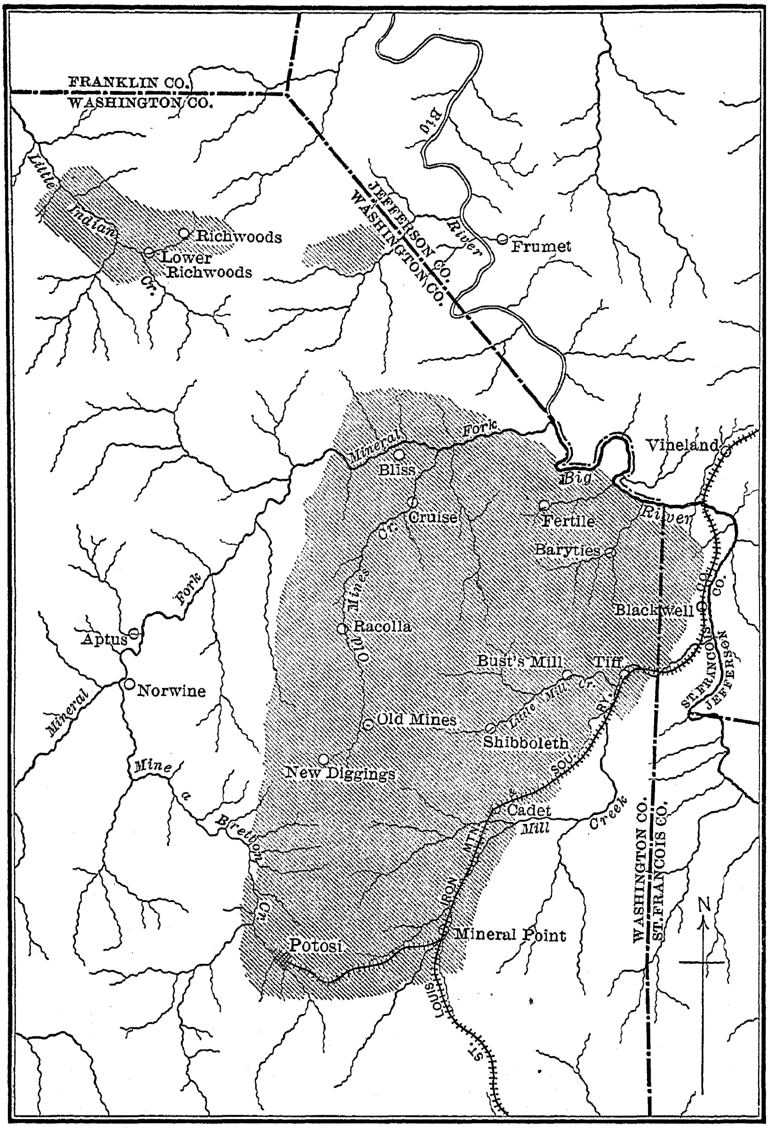


FIG. 1.—SKETCH-MAP OF A PART OF WASHINGTON AND ADJOINING COUNTIES, MISSOURI. SHADED AREAS SHOW BARITE-REGIONS.
(Redrawn from the De Soto, Bonne-Terre, and Potosi Topographic Sheets.)
Scale, 1 in. = 4 miles.

formation. Nearly all the rest of the area is covered to a depth of from 5 to 40 ft. with residual clay containing chert, barite, etc. This over-burden greatly interferes with the study of the

bed-rock, which is exposed only at intervals in stream-beds, in a few barite-pits, and in those lead-mines which have not been abandoned.

I. Geology.

1. *The Formation and Filling of Cavities in Dolomite.*—The Gasconade limestone (of Missouri), of Ordovician age, forms the underlying rock, and is the source of the barite, which is now mostly dug from its residual clay. This rock has been quite uniformly shattered, presumably by dolomitization, since it contains from 40 to 50 per cent. of magnesium carbonate and is full of small, irregular, and non-continuous cracks. Those which have not been greatly enlarged by water are usually less than 1 in. wide and 2 ft. long. Many are very short. They have been shown by diamond-drilling to extend to a depth of more than 600 ft., and are independent of the larger deformation of the strata, which are either level or only gently folded.

A detailed study of the country-rock shows a rather complex series of geological changes. After the first shattering under the obscure conditions of dolomitization, the northern part of the district was raised to the zone of solution. During this time some of the small cracks were connected, forming long branching caves, generally from 2 to 12 ft. wide and from 6 in. to 3 ft. high, and scattered irregularly through the rock at all levels. Several are from 300 to 500 ft. long, and the main channels have a general NW–SE. direction and pitch slightly NW. Since caves formed at this time are found only in the north, a differential elevation of the region may be assumed.

At a time of depression the caves were next coated quite uniformly with chalcedony and then with clear quartz crystals up to 0.5 in. long. At the same time quartz was deposited in the small vug-like cavities which had not been enlarged. There is much quartz as far south as Bust's Mill, and a less amount beyond Potosi.

Ulrich[4] speaks of this quartz as "drusy chert." Some of it occurs in cavities in chert, but much of it is in contact with clean dolomite, and at Ricar, near Racola, two quartz-lined, barite-filled caves are connected by a strip of brecciated car-

---

[4] *Bulletin No.* 267, *U. S. Geological Survey* (1905).

bonate rock cemented with clear quartz, etc. It is therefore assumed that most of the small cavities were formed before the deposition of the quartz, and probably after the formation of the chert.

It is believed that the faulting and extensive fissuring of the district was the next event after the deposition of the quartz. Some of these fissures now filled with barite are free from crystallized quartz and intersect the quartz-lined caves, so this fissuring is clearly later than the quartz-deposition. These cracks are sometimes in intersecting groups like stockworks, but are more often long straight fissures, sometimes traceable for several hundred feet upon the surface and occasionally exposed by mining to a depth of from 50 to 100 ft. without closing up.

In many parts of the field crusts of galena were deposited upon the quartz, and also upon the bare walls of the fissures. Three carloads of copper-ore, mostly as carbonate but with some chalcopyrite, were shipped from Bliss. According to Robert McClay, this ore was found in a single cave next the quartz, and therefore corresponds in age with the earliest galena. No other occurrences of copper were noted in this field. After this deposit of sulphides the remaining space of all of the larger quartz-lined caves and of the fissures was completely filled with white barite, which cleaves into curved plates resembling those of albite. In many places masses of galena are scattered through the barite, which indicates that the deposition of the sulphides continued after the first barite was brought in.

Other localities outside of Missouri show quartz crystals on barite groups, and even pieces of quartz showing casts of barite which has been dissolved by weathering. Since these specimens are durable, and a somewhat careful search failed to find any of them, it seems safe to conclude that most of the quartz was deposited before any of the barite. Winslow reports specimens from Washington county showing a thin deposit of quartz upon galena, but these must be very exceptional. Specimens of barite upon quartz are very plentiful.

What is thought to be a second period of shattering seems to have followed this first deposition of barite. Openings of this second set are scattered among the others and are entirely free from quartz, which would naturally have affected all then-

existing openings before completely sealing so many of the small vugs. The brecciated appearance of the rocks at New Diggings and other places, as well as the angular shape of some of these cavities and their great irregularity and number, seems to preclude the idea that they were formed wholly by subaërial solution. On the other hand, they do not represent brecciation by movements of the strata, and are therefore assumed to have attended the final dolomitization of the rock.

After the second shattering, both sets of openings were subjected to enlargement by water. A few of the first set had the quartz coating loosened, but most of these and some of the second set were unaffected. At the next submergence, the larger cavities in the north were completely filled with barite, and those south of Little Mill creek were coated with dolomite crystals of larger size in the larger openings. The ends and feeder-cracks of the smaller openings were completely filled by this dolomite and occasionally some calcite. Since dolomite would not be readily deposited upon the quartz, and would be easily weathered off, the relative age of the two sets of openings is assumed from the entire absence, in the latter, of quartz showing casts, or found on crystals, of dolomite.

A few cavities still lined with dolomite crystals were seen at Potosi and New Diggings, and dolomite crystals were abundant in the southern part of the field, as shown by the casts upon the back of the barite groups. Since there is no evidence of crystallized dolomite in the northern part of the field, another differential movement may be assumed.

Since the dip of the strata is about parallel to the present surface, and the amount of residual deposit merely varies with the local topography, we cannot explain the difference in the dolomite by differential erosion. Published analyses show no important regional variation in the amount of magnesium in the rock.

In a few places the dolomite was followed by a layer of marcasite, then there was a general deposition of barite upon both the dolomite and marcasite. This barite was never seen completely filling the dolomite crystal-cavities, but is in the form of sheaves of crested crystals. The marcasite was seen at New Diggings, where mining in the bed-rock was still being carried on. In other places it is represented by occasional layers of

hard limonite upon the backs of residual barite groups. More rarely there are shots of limonite imbedded in the barite, showing an overlap in the time of deposition.

The dumps at New Diggings show many specimens, with marcasite followed by barite directly upon the massive dolomite, and a few with barite upon the crystals of dolomite without marcasite. Unfortunately, the original relations could not be determined, since this shaft was full of water. In most other places barite in less-continuous crusts is common directly upon the dolomite. With the barite is often a little galena.

After this second general deposition of barite, the entire region seems to have been raised to the zone of solution. In the north new caves were opened along the old ones which have a floor or roof of the earlier barite. There are also many small separate openings still free from filling, which at times suggest a third period of brecciation, but they are generally continuous, like small caves. In the south, among the cavities containing dolomite crystals are also many of these small openings, and some large caves free from quartz and dolomite. These were probably formed at the same time of general elevation by the solution of some of the dolomite linings or the enlargment of new joints or fractures.

This third set of caves was then submerged, and in them were deposited a few large groups of crested barite, occasionally including cubes of galena or studded with them. This barite is attached directly to the decomposed clay-coated walls of the caves, and seems to be secondary. In some of these caves the walls were coated in places with small calcite crystals. Winslow reports one large scalenohedron of calcite imbedded in barite at Potosi. At Richwoods were seen cross-sections of caves with old barite floors and completely filled with calcite which will cleave into rhombohedra 6 in. on a side. There are other caves now above water-level, with the old barite floor thickly strewn with eroded scalenohedra of calcite, and with the roof studded with smaller barite groups and calcite crystals. These caves are apparently free from stalactites, and the calcite and barite have therefore been deposited below water-level.

At a slight depth below water-level the barite groups seem to be finished by a layer of very small needle-like crystals of barite. They are still later coated in places with a stain of

limonite, which does not collect on the other minerals. This result is probably caused by the attraction of barium sulphate for ferric salts, which makes it necessary to remove the iron from solutions before precipitating the barium in quantitative analysis.

At a greater depth, at New Diggings, there are recent contemporaneous deposits of galena, clear ruby sphalerite, and barite upon the older barite. This last barite is in small, pure white sheaves and seems to be still growing.

The weathering-surface of the bed-rock intersects many of the caves, which are then filled by a recent deposit of fine red clay. This obscures the relationship of the minerals in many of the exposures of bed-rock. It is, of course, the last of the cave-deposits.

It is possible that the second set of fractures (those lined with dolomite) may have been formed before the first introduction of barite into the fissures. In favor of this is the failure to find any dolomite upon barite, while barite is frequently found upon dolomite. When they are associated, the marcasite is later than the dolomite, and generally older than the barite.

As to the first reason, dolomite is so soluble that it would during subsequent changes be easily removed from the barite. It would also deposit upon the abundant massive dolomite or its own crystals rather than upon the barite, and if the first set of fissures were entirely filled with barite no dolomite could be deposited upon it. All barite which showed casts of dolomite was in the form of sheaves or incomplete fillings of openings, and probably later than the first barite deposited. As to the second, there are in immediate association specimens without the marcasite and others without the dolomite crystals. This I am unable to explain.

As evidence that the fissures were filled with barite before the second brecciation of the rock, there are straight and sometimes branching fractures entirely filled with clean massive barite in contact with fresh unaltered walls, and in their immediate vicinity many irregular cavities lined with dolomite and only partly filled with barite in scattered sheaves.

Besides those fractures, which I have seen, Mr. Robertson[5] describes such a fissure near Potosi intersecting " absolutely

[5] *Report of the Missouri Geological Survey*, vol. vii., p. 679 (1894).

unaltered" rock, which was followed to a depth of 60 ft. The rock was full of quartz cavities, but dolomite was not mentioned. On page 678 he mentions another fissure near the Bugg shaft, near Potosi, in rock with dolomite-lined cavities. This fissure contained lead and barite, but had been affected by later solutions. It was from 0 to 10 ft. wide, more than 112 ft. deep, and 1,300 ft. long. Such sharp fractures would not be likely to form in a rock so full of cavities poorly cemented with dolomite, and containing quite large and irregular openings. On the other hand, if fissures filled with barite were already present, they would resist the shattering and would obstruct the dolomite solutions, and so tend to prevent intersections of the two sets of openings. No intersections were found, but there were only a few exposures for study. Many of these, however, showed dolomite openings near the barite fissures.

It is also difficult to understand how these fissures could be so completely filled and the other sort of openings receive only scattered deposits if both existed at the time of the introduction of the barite. Even those fissures that have been enlarged by solution show slabs of barite which evidently once completely filled a smaller opening. In the north, where there are good exposures, the presence of a few scattered barite sheaves of later growth upon the walls of reopened caves with barite floors and feeders shows clearly that barite has been deposited at more than one period of submergence, with at least one elevation of the region intervening. Specimens were also found showing barite sheaves, probably of the second period, growing upon the clean outer surface of drusy quartz, with apparently later barite continuing the growth over upon the iron-stained back of the quartz. This proves an extensive interval of solution between periods of barite deposits. The barite fissures are found over an extensive area, and are clearly independent of the somewhat local crystallization of dolomite.

It seems more probable, therefore, that the first main introduction of barite was, as stated, before the formation of the dolomite-lined cavities.

2. *Summary of the Geological History of Washington County.*— In Archæan time the St. Francois mountains of Missouri were formed of red granite and porphyry, with dikes of a basic rock.

During a long pre-Cambrian period they weathered to a low irregular dome with outlying prominences. During this time the veins of iron-ore were formed, and they were otherwise affected by circulating solutions. During Cambrian and later times they were probably never wholly submerged, but sediments were deposited upon their flanks and in the valleys, all dipping roughly parallel to the old land-surface. The first sediments are of land-wash and contain in places the bedded iron-ores, which were succeeded by limestones with local beds of sandstone and many unconformities and other evidences of oscillation in level. Washington county has been land-surface continuously, or only submerged for brief intervals, from Carboniferous or possibly earlier time to the present. During this period there was much erosion, and the outlying granite knobs of the Archæan mountains were exposed in southern Washington county.

Some time after the deposition of the Gasconade limestone, the chert was formed in it. Later, the rock was rather minutely fractured over a wide area, and the region along Mineral fork and Indian creek raised sufficiently to permit the formation of many small limestone caves. The entire region was then submerged and all existing openings lined with quartz. There was next a rather severe fissuring and faulting, followed by the introduction of barite with more or less lead, copper, and iron sulphides and probably some zinc sulphide.

This period was followed by another shattering, which was less pronounced in the northern part of the county. Many of these new fractures were soon enlarged by water, and near Potosi the country was lowered and the openings lined with dolomite crystals, which were followed in a few places by a layer of marcasite, and generally by barite in scattered sheaves, which usually also formed upon the marcasite. After the marcasite was deposited, the lowering seems to have become more general, and the few caves in the northern part of the field were rather completely filled with barite.

A general elevation followed this, and the large and small caves were formed throughout the barite-field. Upon the next submergence, a little barite was deposited in them in isolated groups along with more metallic sulphides. Calcite in double-pointed scalenohedra was also deposited at this time or soon

afterwards. When in contact with oxidizing solutions, the free surface of the barite became coated with limonite, which is usually the last deposit from solution.

At intervals ever since the first introduction of the barite, the massive dolomite has been dissolved by weather, and the barite and galena and other insoluble matter has accumulated in residual deposits. The fine part of this is generally washed into those caves which came to the surface of the rock.

3. *Comparison with Other Districts.*—All of the barite of Washington county has clearly been deposited in free open spaces. It is all very white, rather soft, and either in poorly-cleavable masses from completely-filled openings, or in sheaves with ridged surfaces. When attached by the side, these groups are spindle-shaped, but more commonly they are attached at one end and are roughly heart-shaped. They weigh up to 20 lb. each, and cleave like the massive barite. No one has seen or heard of clear or translucent barite, or any of the common tabular crystals, occurring in this county.

At Morrilton, Franklin county, just beyond this field, is one nearly-vertical cave, 60 by 100 ft. in section, filled with very unctuous clay containing groups of beautiful blue, square, tabular crystals of barite, sometimes weathered white on the edges and sometimes attached to pieces of chert. In this clay were also found a few large barite stalactites covered with fine, colorless, tabular crystals of barite. Aside from this occurrence, all the barite seen in the central counties of Missouri was in the form of "slabs," which had completely filled flat caves, or groups of vertical fissures. Winslow mentions blue barite crystals upon the dumps of the Virginia mine in Franklin county.

In the fluorspar-region of Crittenden and Caldwell counties, Ky., there are some few veins of barite containing strontium, and mixed with fluorite and white calcite. At the time of my visit no barite was being mined there. Some details of this occurrence have been given by F. Julius Fohs.[6] A complete account will soon be published in another bulletin by the same author, which will include an account of the barite of central Kentucky.

Near Cleveland, Tenn., and in western Virginia, most of the

---

[6] *Bulletin No.* 9, *Kentucky Geological Survey* (1907).

barite fills groups of fissures and not caves in the Knox dolomite, and there has been a great deal of metasomatic replacement of the wall-rock. At Honaker, Va., there are some small masses of sulphur in the barite, as well as iron sulphide. At Sweetwater, Tenn., in the center of the belt, some barite has incompletely filled fissures, as in Washington county, Mo. It was, however, in the form of uniform layers or crusts upon the walls with imperfect, small, tabular crystals on the surface. Here also was a much larger amount of earlier marcasite and some contemporaneous fluorite.

Further east in North Carolina and Virginia the barite is quite translucent, and is said to occur in distinct fissures in gneiss, quartzite, or sandstone. At Evington, Va., the barite seems to have replaced parts of lenses of marble in a micaceous schist.[7] The mines were so largely abandoned that all the exact relations could not be worked out at the time of my visit.

4. *Origin of the Barite.*—The origin of the barite has been discussed only by Mr. Winslow, of the Missouri Geological Survey. He thinks that the barite, together with its associated lead and zinc, has accumulated through the agency of descending solutions, gathering the barite from the weathering of the dolomitic rock. In favor of this hypothesis is the presence of appreciable, although very small, amounts of barite in the dolomite (from 0.001 to 0.005 per cent.); the assumed absence of definitely-proved fissures extending to great depths; the long period and considerable amount of denudation; the evidence of the Flat River lead-mines; and the stalactites of barite in Franklin county.

On the other hand, Winslow himself says that both the porphyry and the granite in the southern part of Washington county and beyond show barite in visible amounts. He especially mentions the porphyry at Hogan, Iron county. It seems safe, therefore, to assume that, in common with most igneous rocks in which it has been determined, these rocks contain barite in much greater proportion than does the dolomite. (No printed reports of the quantitative determinations of the percentage of barium oxide in these rocks can be found.) These Archæan

[7] Watson, T. L. Geology of the Virginia Barite-Deposits, *Trans.*, xxxviii., 717 to 719 (1908).

rocks underlie the entire region, and would, therefore, prove an adequate source for the barium of the barite, which they have brought from still deeper regions.

Clarke's tables of rock-analyses[8] give averages of from 0.04 to 0.06 per cent. of barium oxide for shales, sandstones, and igneous rocks, and the analyses of two groups of samples of limestones showed no barium at all. Since even the Potosi limestone has only less than 0.005 per cent. of barium sulphate, one would expect that, according to the descension hypothesis, barite should be most abundantly present in fissures or replaceable limestone in shale- and sandstone-districts, which have been subjected to long periods of weathering. It seems best to assume, therefore, that the calcium and magnesium carbonate in which the barite is generally found simply serves as a precipitating-agent in which cavities could be easily formed.

All of the quartz in Washington county is older than the barite. The best explanation of this obvious quartz-cementation requires alternations in the flow of the water. In the downward course it dissolves some of that quartz which is combined with soluble bases, and while slowly rising it deposits quartz from cooling solutions of silicic acid. Some barite, if enough were present, would naturally have been deposited in the small vugs or as concretions in the limestone at one or the other of these stages, for alternate slight solution and deposition always make large minerals grow at the expense of small ones of the same sort. This growth is also due to the greater solubility of smaller particles.

All the barite is deposited in the larger or more continuous openings. In a general circulation, according to Van Hise, these are channels of rising currents. At any rate, they are the channels of rapid movement At a time of elevation of the land, the rapid movement would be downward, but the solution could not then become saturated with barite at the suface, and there are no signs of the deposition of the barite in caves whose walls were being dissolved, a result which would be noticeable on the backs of barite sheaves. At a time of depression, stored solutions would flow rapidly upward and finally into old caves, etc. This is clearly the stage in which the barite was de-

[8] *Bulletin No.* 168, *U. S. Geological Survey* (1900).

posited. (The solution may have received its barite in the belt of weathering, but its saturation would then indicate stagnation below.)

In Franklin county, just north, are many definite veins of galena and barite in fault-fissures. At the Virginia mine[9] a fault striking N. 20° W., carrying usually from 18 to 24 in. of barite undiminished below, was opened to a depth of 480 ft. and a length of 1 mile. Here is, therefore, at least one fissure which undoubtedly extends to sufficient depth to reach the igneous rocks. There are many other faults, probably also deep.

In Tennessee there are single fissures, and especially groups of intersecting fissures, in which there has been much metasomatic replacement of the walls. Until more work has been done upon the conditions of solubility of barite, it is unreasonable to suppose that solutions saturated with barite in the belt of weathering could descend through dolomite all the way, and at greater depth be able to dissolve more dolomite and precipitate barite in its place. It may, therefore, be concluded that these fracture-zones extend to a deep source of barite. In Missouri, the amount of denudation has been but a fraction of that in the Tennessee valley, especially if that accomplished previous to the deposition of the quartz be deducted. Therefore, in Missouri the tops of the barite-veins are found, while in Tennessee the bottoms are seen. The differences are similar to those between the tops and bottoms of gold-veins. Hence the similar Washington county fracture-zones probably extend to equal depths.

The difficulty of the metasomatic replacement of a rock by a totally different chemical compound and by solutions already saturated with the same rock-constituents by a long downward course through the rock, applies with even greater force to the Flat River lead-ores, which generally replace the country-rock most abundantly in the dolomite stratum nearest the Archæan rocks beneath.

The barite stalactites at Morrilton, and others mentioned by Winslow as occurring in Franklin county, have clearly been formed by descending solutions, but these may have obtained

---

[9] *Report of the Missouri Geological Survey*, vol. vii., p. 696 (1894).

their barite from the weathering of earlier veins filled from below. The stalactites were afterwards coated with perfect tabular crystals. Since all the barite crystals of Washington county are of the crested type, and the massive barite is not stalactitic, it is quite apparent that the Washington county barite was deposited under different conditions, though not necessarily by rising solutions.

Most of the barite of Missouri at all stages of deposition is contemporaneous with more or less galena, which has the same order of solubility. It is seldom, if ever, contemporaneous with calcite, and rarely with dolomite. This indicates a separate source or time of solution of the carbonates and the barite. The carbonates in Washington county would naturally come from the zone of solution.

In Kentucky, the barite is contemporaneous with white calcite, but there it is also clearly contemporaneous with fluorite, and obviously in fissure-veins derived from below, and associated with igneous dikes (0.06 per cent. of barium oxide, BaO).

Van Hise [10] says that fluorite weathers to calcite. Therefore the presence of a little fluorite in the barite of the Knox dolomite and of much fluorite in the barite of Caldwell county, Ky., proves that the veins in these carbonate-regions were filled by solutions from below. Unfortunately, no special search was made for fluorite in the barite of Missouri, because this association in the other regions was not known to me at the time the field-work was done. It is certainly not common. I have observed argentiferous galena in a gangue of massive barite of Missouri type, filling fissure-veins in granite and porphyry, in the White mountains of New Mexico. The walls of these veins had been extensively altered to sericite, and they were obviously filled from below. Similar instances of barite coming from below and quoted in the literature could be multiplied almost indefinitely. Thus there is nothing strange about the process.

The same dolomite covers a wide area of Missouri. In many places it seems to lack an excess of unfilled caves and fissures, but in Washington county at least there are a great many of even the oldest quartz-lined openings entirely without barite.

[10] *Treatise on Metamorphism*, p. 373 (1904).

Still, the barite occurs in decidedly limited patches, often long and narrow. One man's field may yield 2,500 tons of barite per acre from the residual soil, while another just across the road will have practically no barite. This seems to be the most decisive proof that the barite came in through fissure-zones from below, rather than from the uniform dolomite present everywhere.

Much barite and galena are now disintegrating in the residual soil, and some barite doubtless goes into solution and is precipitated immediately below. This probably accounts for the apparent present continuance of the formation of barite crystals. Depending upon its fineness, barite is soluble to the extent of from 23 to 40 parts in 10,000,000 parts of water. It is, however, somewhat readily soluble in strong solutions of carbonates in the presence of carbonic acid. Its solubility under these conditions increases rapidly as the pressure of carbon dioxide increases, which would tend to prevent the precipitation of barite from descending solutions. The influence of the increasing amount of magnesium carbonate presents an unknown factor which makes chemical speculations idle. Barite, however, does not suffer solution by oxidation at the surface and precipitation by reduction below the zone of solution, as does galena.

The evidence, therefore, seems conclusive that the barite originally came from below. The barite-bearing granites and porphyries of Archæan age are not very deeply buried in Washington county, so the faults certainly extend down into them and they afford a convenient source. In pre-Algonkian time these rocks were greatly affected by solutions, and the visible barite may have been then concentrated or brought in from the basic dikes intrusive in them.

Since all the igneous rocks suffered a long period of weathering before the Cambrian and later sediments were put down, magmatic water, etc., could have had no immediate part in the introduction of the barite, which, with the Washington and Franklin county galena, must then have been brought up by rising trunk-currents of the circulation of meteoric water, some of which fell upon the igneous rocks of the St. Francois mountains, which were always free from Palæozoic rocks and higher than the Washington county sea- or land-level.

5. *Present Condition of the Barite.*—As the dolomite became dissolved the barite accumulated with the clay, flint, and other residual material. Commonly the barite is somewhat concentrated at a depth of from 2 to 6 ft., probably the lower limit of the disturbance of the soil by frost and other agencies. Above this, the clay contains an unusual amount of flint or chert uniformly scattered through it.

Beneath the first bed of barite the clay seems more undisturbed and the barite is scattered irregularly through it. Most of the slabs of barite have fallen over and are now horizontal, and there is likely to be a little more barite immediately above the bed-rock. The upper part of the weathered dolomite is soft and crumbly and is often called sandstone. Some of the larger slabs of barite are dug from this until it becomes too hard. Near the bed-rock the barite is quite hard, more nearly translucent, and contains iron-stains only upon the surface.

Near the surface the barite is commonly not much softened, but thin layers of limonite are deposited in the cleavage-planes, along which the barite breaks up. When very dark and purplish this barite is sold at half price as No. 2, or "battle ax," on account of the difficulty of bleaching it. There is a little which is deep yellow all through. It was said that this stain was not due to iron and could not be removed by sulphuric acid in the bleaching-process, even by those mills which grind before they bleach. It cannot therefore be sold. Laboratory-tests show that the stain is only iron and clay, and perfectly soluble in rather strong sulphuric acid. Frequently, quite a lot of barite of walnut-size is found near the bottom of the flint-zone.

In places the lumps of barite adhere to a good deal of chert which probably once projected from the walls of the caves. Upon other specimens are shells of hard, dark limonite, called "metallic iron," and representing weathered marcasite. Both the flint and the iron can be chopped off with a hatchet. The quartz crusts, called "mineral blossom," are so irregular and so frequently included in the barite that they cannot be chopped off, and reduce the grade to No. 2, because the resulting grit is especially objectionable in the barite floated for pigment. Where there is much mineral blossom the barite is therefore not mined. A little barite which probably once contained

strontium sulphate is now very soft and dazzlingly white. It is sticky in the grinding-mills and cannot be sold.

In each particular locality the miners have more or less reasonable theories as to the occurrence of the barite along certain leads. At Bliss there certainly are long straight bands of barite running NW. and representing old cave-fillings. These bands are only from 20 to 50 ft. apart, or closer than the caves, due to the great vertical concentration caused by the solution of the limestone. In most places the barite seems to have been derived from groups of caves or fissures, and it is rather uniformly distributed, though often irregular, crooked, richer streaks can be followed through the elongated patches. The method of mining by rows of pits usually accounts for the idea of leads. There is also no uniform ratio between the amounts of barite and of galena, for some of the best barite-patches yield very little galena, and *vice versâ*.

II. The Mining Industry.

1. *Amount of Barite Available and the Output.*—The entire area is very irregularly supplied with barite, which is commonly mined in patches from 10 to 15 acres in size. The longest single patch is at Mineral Point, where the crest of a low ridge has been mined almost continuously from 50 to 300 ft. wide for a distance of half a mile. The barite, or "tiff," as it is called by the miners, is mostly mined from cleared farm-fields, which showed a little barite upon plowing and were accordingly prospected. The patches are therefore most frequently upon the slopes of the hill-sides, where the barite is likely to be exposed. There are probably several good patches in the woods which have not yet been found, although the country has been prospected for more than a century.

James Long, of Potosi, the oldest and largest shipper of barite, estimates that of the entire area of about 100 sq. miles, one-tenth is workable barite-land. The best fields have produced from 2,500 to 4,000 tons per acre, which is probably only about one-half the barite they contain. The average field, under the present system of mining, yields about 600 tons per acre to a depth of 8 ft., or about 100 lb. per cubic yard, but it takes many years to produce this amount, and very few fields have been anything like exhausted as yet. Upon this basis of

estimate, the probable total yield, without a change in mining-methods, will be about 4,000,000 tons, or more than a century's supply, if we count new discoveries equivalent to the amount already taken out.

Every little country store accepts barite in exchange for supplies, but it is all shipped through half a dozen contractors. The total amount shipped was given as 33,046 tons. Nearly all of this was first-class barite, and sold for $4.35 per ton at the shipping-point, or $5 at St. Louis. In 1905, the price was reduced to $4.10, at which it now stands. The royalty to the owner of the land is quite uniformly 50 cents per ton, but it is 80 cents on one especially good field, and as low as 15 cents when very far from the railroad. The miners get as much as $2.75 a ton when much cleaning is necessary, but usually from $2.20 to $2.50, and only $1 a ton at Richwoods, from which the wagon-haul costs $2.50. The haul usually costs from 25 to 50 cents a ton, and the contractor gets the rest. Farm-land containing barite in some of the fields sells at $35 an acre, which is but little different from its farming-value. Wild land, from which the best timber has been cut, is worth only from $2 to $8 an acre. Some land next to the railroad-siding, and all containing a little barite, was bought at $60 per acre. One 10-acre field near Potosi has yielded a royalty of $200 per acre each year for 10 years, and a neighboring field, which is supposed to be just as good, is still farmed and held in reserve. About 400 miners are supported wholly by the barite industry, besides those engaged in hauling, etc. So far as I could ascertain, these men do practically no farming, although it is otherwise stated in the books.

2. *Mining-Operations.*—A little barite is still produced from mines extending below water-level into bed-rock, but only when associated with enough lead to pay expenses. The small scale of this work and the inexpensive plant form a striking contrast to the great mines at Flat River, a few miles east. Each shaft usually follows a vertical fissure until a cave is reached, when a drift only 3.5 or 4 ft. high is driven along the little horizontal stringer of galena and barite, which is commonly less than 6 in. thick and carries about 25 per cent. of lead. Deposits having a cave above or below are naturally preferred, since the cost of blasting is reduced, and in addition

large crystals of galena frequently occur upon the surface, from which they are easily cobbed off. The little ore-streaks are followed as long as they nearly pay expenses, since they sometimes lead to large and profitable bodies of ore.

Most of the barite is dug from the residual clay. The common plan is to sink a pit, 3 ft. in diameter, to a depth of from 6 to 9 ft., to the first lean clay below the upper barite-layer. The miner then selects the richest side of the shaft, which he digs out beneath the upper clay, which is rather barren for from 3 to 6 ft. below the surface. In this way he "drifts" as far as he thinks is safe, usually from 4 to 8 ft. Up to this stage all of the clay, flint, and barite loosened is hoisted in a small bucket by a primitive hand-windlass; but the miner usually sorts out the barite, which he distinguishes from flint only by its greater weight, since it is completely covered with tough red clay. When the first drift is finished all the waste from the new drifts is shoveled into it, and the miner digs other drifts wherever the ground looks good, so that the working soon resembles an inverted mushroom.

When the first hole is finished, another is started in the direction that looks best, at such a distance that the safe drift backward will just connect with the old workings and give a little ventilation. The second hole is exhausted as before and a third one dug beyond. This plan produces a more or less regular row of pits, and gives rise to the theory of leads. When a lean place is struck the lead is said to have given out, and another row of holes is started, until finally they are scattered over an old field like the squares of a checker-board, and perhaps one-half of the barite is removed.

I have heard of no accident attending the mining, and since many of these holes remain open for years, in spite of the rain, it seems probable that the length of the drift is not limited by safety, but rather by the distance reached by daylight, and the greater convenience of working near the shaft, which about balances the labor of sinking a new pit with the ordinary amount of covering. Owing to the tendency to imitate, the size is the same whether there is 2 ft. or 6 ft. of barren cover.

When the upper layer of barite has been exhausted, or the field is very rich, the pits are sunk deeper. In summer, when

the ground is cooler than the air, the ventilation is poor, so that the pits are seldom deeper than from 15 to 18 ft. In winter they go to bed-rock, or sometimes more than 30 ft. These deeper pits are often worked like the others. The more intelligent miners generally drive drifts from 4 to 6 ft. high for 8 or 10 ft. in three directions, as shown in Fig. 2, and connect the ends so as to leave pillars. The roof is then caved or stoped upward and the waste left on the floor. As the space fills up and the old workings are approached the area is de-

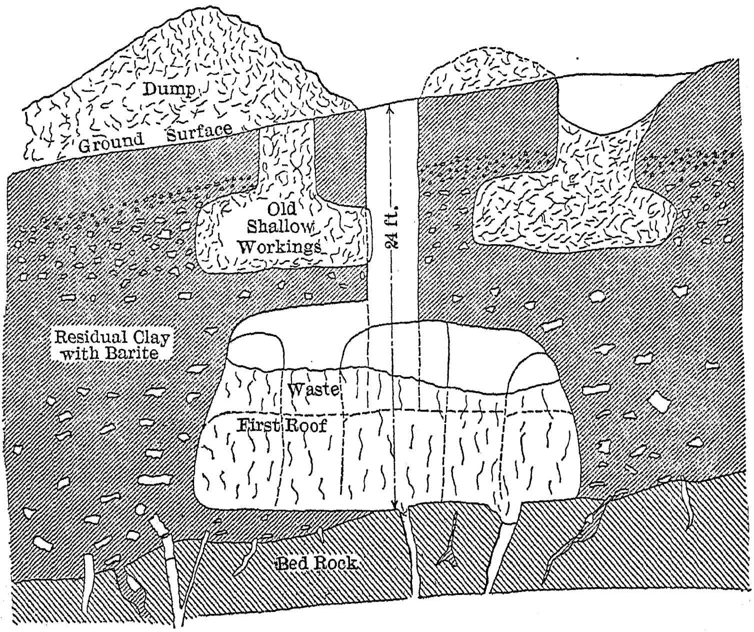


Fig. 2.—Cross-Section of Deeper Barite-Workings.

creased. The new pit is sunk so close to the last forward drift that but little drifting is required to make the air-connection. The connection-drift is often stoped from the second shaft. By careful work four-fifths of the area covered by the row of workings can be mined; but since the drifts are stopped wherever the ground looks poor, and the pillars are larger than the drifts, and the stopes decrease in size upwards, more than one-half or two-thirds of the ground is seldom mined. Since one row of pits is driven without regard to the others, and a good deal of pillar must be left between rows, at best only a small percent-

age of the barite is recovered. Indeed, so much is left that I saw no field of pits which was said to be worked out.

As the price paid for mining increases, men will dig around among the old workings, in spite of the annoyance of striking old drifts, even where the pits have been sunk to bed-rock. This shows that much barite is lost between the pits. There is also a loss due to the irregularities in the surface of the bed-rock and the caving-in of unfinished holes by rain. Since the miner leaves the poorer parts, the loss of barite is not as great as the percentage of ground unworked. In a field rather carefully worked to bed-rock it was roughly estimated that half the barite was lost.

While one man digs the barite, his partner or wife stays on top to work the windlass, and with a small tomahawk-shaped hatchet cleans the barite of the clay, the worst iron-stain, and any flint that may adhere to it. In summer a rough sun-shade of branches is built.

Most of the clean barite is in lumps, weighing from 3 lb. (large fist-size) to 20 lb., and the finer stuff is thrown away, except near the railroad. Here the small pieces at the top of the barite-layer are piled upon the ground, where the weather loosens some of the clay. This barite is later cleaned in a tumbling-barrel or "rattle-box," and shipped as second grade on account of the larger amount of iron-stain upon it. The rattle-box, shown in Fig. 3, is a strong box, 18 by 30 in. by 10 in. deep, with a slat bottom and the ends thickly studded with strong spikes driven through from the outside. On the sides are uprights of 2- by 4-in. pieces, forming legs 1 ft. long and handles 3 ft. long. From 100 to 150 lb. of small and thoroughly dried barite is shoveled into the box, which is then rocked upon its feet by two men. When the end of the box strikes the ground, the barite is thrown forcibly against the spikes until all the clay is knocked off and falls through the slats.

The miners receive from $1.60 to $2.75 per ton for cleaned barite, and by working 6 hr. per day a pair will usually earn from $6 to $7 in 5 days. Since they can live a week upon this amount, they seldom earn more than $8 in a single week, and when a rich spot is struck they spend the time loafing and drinking up the excess earnings. When the ground is lean they work longer.

The miners are usually the most ignorant "gumbo French," and speak no language but their own French dialect, and therefore resist any change in the system of mining. Their scheme has the advantage of all "gopher" mining, that the minimum of barren material need be handled. When the over-burden is thick, this method is probably as good as any, if controlled by an efficient superintendent who would see that the pits are regularly spaced, except where there are real leads. The drifts could generally be higher and longer with advantage.

The Point Mining & Milling Co. installed a pneumatic cleaning-pick and a mechanical washer, and tried to buy uncleaned barite at a lower price. By having one top-man for

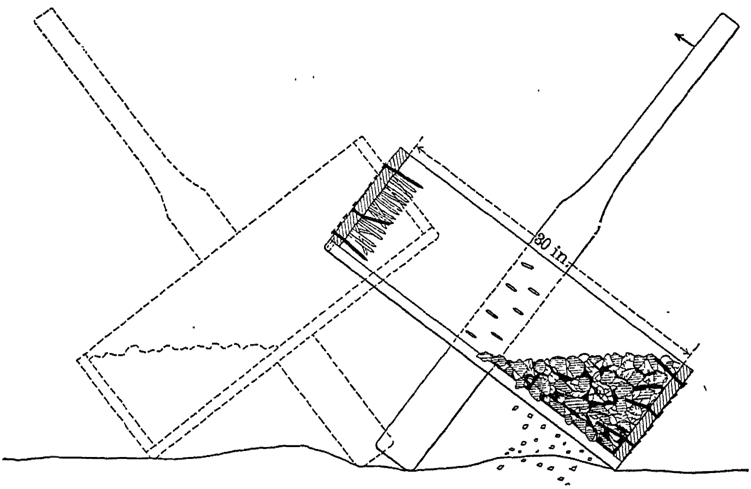


FIG. 3.—"RATTLE-BOX" FOR CLEANING FINE BARITE.

two diggers, the miners earned more money upon this basis; but they did not want to work in threes, and, according to the old arrangement, the top-man has plenty of time to clean the barite. Therefore this machinery is not now used.

Some of the better farming-land will not be released for barite-mining, since the pits and dumps ruin the land. Occasionally the top layer of barite in a small field is mined out rapidly and more systematically, and the miners are required to put the waste material into the pit immediately behind. When the larger stones are put in first, this mining improves the land.

In two or three cases fields having but little stripping were

mined by advancing a trench 6 ft. deep across them sidewise. This method gives much better recovery, and affords a good opportunity of placing the best soil on top, and would be used oftener, except that it is hard to get the miners to work for day's pay, and constant supervision is required.

One large company, now bankrupt, wasted a good deal of money upon a traction-engine and wooden-rail tramways for hauling, in competition with the Missouri mule in his own home. At the time of my visit uncleaned barite was mined by day-labor, the clay washed off in a patented trommel-washer, and then the iron and flint chopped off by hand. This system gave a considerable saving, even though the work was inefficiently managed. A little saving was probably due to the greater richness of the newly-opened field, but the method is hopeful.

At the Duffy farm a big company undertook to work the barite hydraulically. The exact arrangement of the sluices could not be determined. On the gentle hill-side above the reservoir is a pit 40 by 80 ft. and about 10 ft. deep to bed-rock. Everything seems to have been well constructed, but the method soon failed for lack of water and of fall to remove the tailings. A little trouble also arose from the depressions in the bed-rock. There are but few places in this field where there will not be trouble with tailings. So it is apparently better to abandon sluices, hydraulic the material over a grizzly, pick out the barite, and remove the tailings by some sort of elevator. The tough clay is hard to handle by water, and labor is so cheap that labor-saving machinery may not pay.

The engineer of the Point Mining & Milling Co. was thinking of mining the residual soil with a steam-shovel, running it through a trommel to shake loose the clay, and carrying the refuse to the dump by belt-conveyer, from which the barite should be picked by hand. This system would be especially adapted to the thicker level deposits. The daily output would be very great, and a large crew of pickers would have to be organized, and there would be some expense in handling the tailings. Since much of the land will yield from 125 to 250 lb. of barite per cubic yard of soil, and the cost of mining uncleaned barite by present methods is about $1.50 per ton, this system shows a margin of from 10 to 20 cents per cubic yard for handling by steam-shovel, which is not especially attractive. In

localities where water is available, it might be better to treat the soil in a modified log-washer, and pick out the partly-cleaned barite.

In most places the best method of mining seems to be to strip the barren cover with plows and scrapers, and mine the rest with pick and shovel, as is done in Tennessee and in central Missouri, where the deposits are smaller and richer. It had not been tried in Washington county previous to 1905.

III. Preparation for the Market.

1. *The Barite-Mill at Mineral Point.*—Previous to 1905 most of the barite of Missouri was prepared in the mill of Nulsen, Klein & Krausse, and some by J. C. Fink Mineral & Milling Co., both of St. Louis. The former has a department for making floated barite, resembling that of the new mill at Mineral Point, but a second-grade pigment is also made from unbleached selected barite. These mills are not in Washington county, and it seems better to omit a discussion of them, since they are similar to the many other mills which have been previously described.

During 1904 the Point Mining & Milling Co. invested about $150,000 for barite-land and what is claimed to be the largest wet-process mill in the United States. Its capacity is about 35 tons of finished product per 24 hr., or, allowing for delays, about 10,000 tons per year. The mill was designed by W. R. Macklind, who has kindly given me the following data.

A diagram of the operations at this mill is given in Fig. 4.

The barite is weighed upon a platform-scale and shoveled into the ore-house, which has a horizontal air-drill with chisel point for cleaning the flint and iron from the barite when this is not done by the miners.

From the ore-house the barite is shoveled into an 11- by 16-in. Blake breaker below the floor, which reduces it to pieces of 1-in. size. It is then elevated to a 30-ton bin at the top of the room, whence it is taken to three modified log-washers in series, in which nearly all of the clay is removed.

The washed barite is wheeled across the floor to two Macklind "slip"-mills or modified *arrastres*, through which a large stream of water flows, and floats the fine barite over the edge of the pan. This milky water is raised to the top of the mill

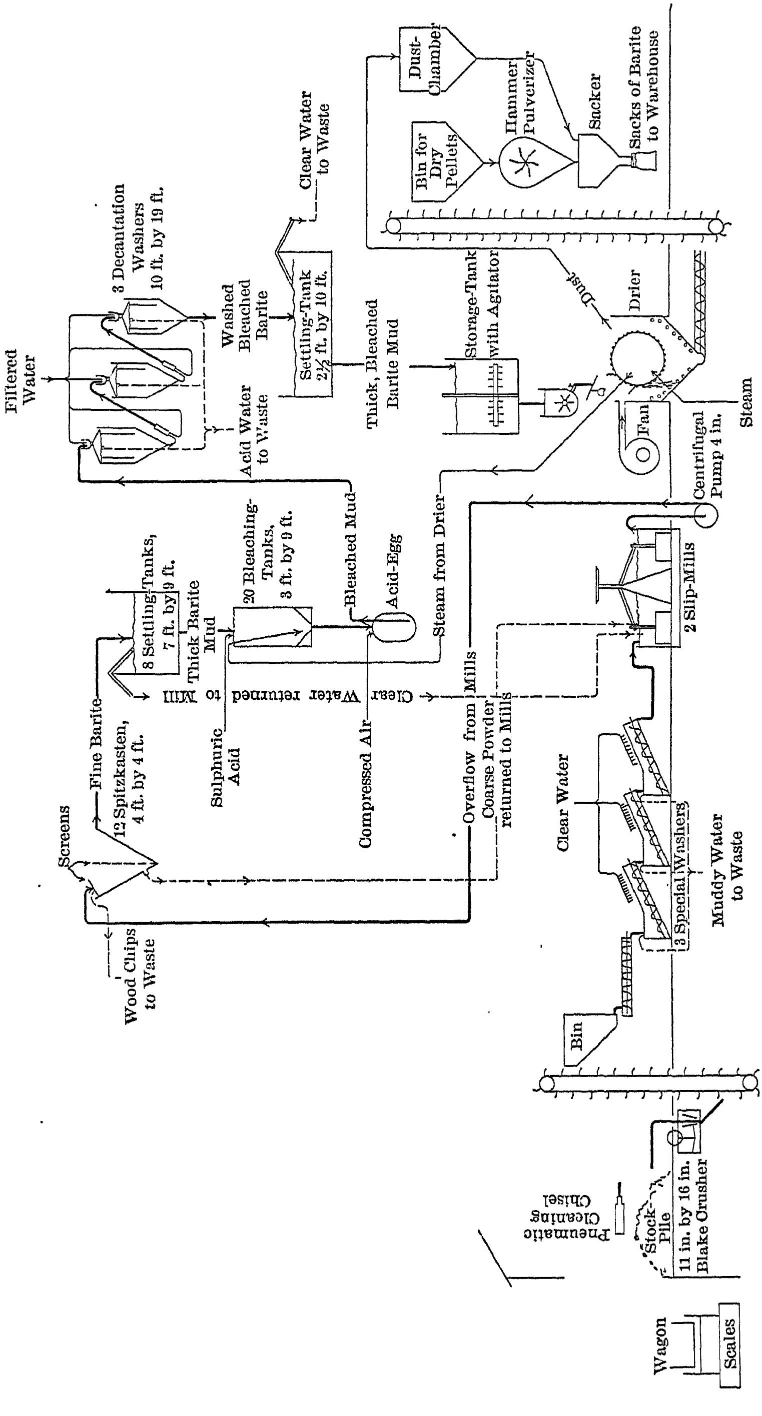


FIG. 4.—DIAGRAM SHOWING THE PREPARATION OF BARITE IN THE MILL OF THE POINT MINING & MILLING CO., MINERAL POINT, MO.

by a 4-in. centrifugal pump, and fed through a small screen into 12 iron *spitzkasten*, 4 by 4 ft. in size. The oversize is drawn off through spigots at the bottom of each box and returned to the slip-mills. The fine barite overflows opposite the entering stream into one of 8 settling-tanks, 7 by 9 ft. in size. The clear water is siphoned from these tanks and returned to the slip-mill. The thick cream-colored barite mud, which contains but 25 per cent. of water, is drawn off through 4-in. plug-valves and a rubber hose to one of 20 bleaching-tanks on the floor below.

The required amount of 66 per cent. sulphuric acid is added, and the mass agitated and kept at the boiling-point by a jet of steam introduced through a lead pipe. When the barite becomes white it is drawn off through a plug-valve in the bottom, and raised by an acid-egg to the first of three Macklind continuous decantation-washers, in which nearly all of the free acid is removed. From the bottom of the last washer the barite is drawn off to one of two settling-tanks, 2.5 ft. deep and 10 ft. in diameter. Here as much clear water as possible is siphoned off, and the barite let down into a storage-tank provided with an agitator to keep the barite mobile, since it contains but 15 per cent. of water.

From the storage-tank the barite flows to a Macklind continuous automatic drier. A screw-conveyor collects the pellets of barite below the drier and delivers them to a belt-elevator supplying a bin above a Williams hammer-mill which pulverizes these pellets. The product is packed in 100-lb. duck sacks, and stored in a warehouse of 3,000 tons capacity.

The mill is driven by an 18- by 36-in. 200-h.p. Fulton-Corliss engine, running at 75 rev. per min., and supplied with steam at 100 lb. per sq. in. pressure, by two 72-in. by 18-ft. return-tubular boilers. There are a feed-water heater, duplicate feed-pumps, etc. A 35-kw. direct-connected Westinghouse unit supplies current for lighting the mill, and a Curtiss belt-driven compressor supplies the compressed air.

Besides the superintendent and office-man, there are 12 laborers at $1.50 a day on the two shifts, two slip-mill men at $2, one bleacher at $2, two engineers at $2.50, and two firemen at $1.75, all working 12-hr. shifts. At the time of my visit the mill had not been operated long enough to get reli-

able data of costs, and details of operation were being constantly improved. The cost of manufacture is somewhat high.

Much of the machinery is standard and need not be described. Each unit of the washer consists of a 12-in. screw-conveyor, 10 ft. long, inclined at an angle of 20°. The housing of the conveyor is arranged to hold the water-level high enough to cover two-thirds of the length of the screw. Above this the barite, while being stirred by the screw, is sprayed with about 25 jets of water, $\frac{1}{8}$ in. in size, under a 45-ft. head. The edge of the screw-conveyor has a V-shaped notch, 1.5 in. deep, every 4 in., which allows a large part of the fine material containing the most clay to flow back again and be rewashed. At the lower end of the conveyor a strong jet of water washes the mud and a little of the finest barite over the edge of the housing. The barite is fed upon the lower end of the first conveyor and discharged from the top of the last one. These washers have a capacity of about 120 tons of barite in 24 hr., and consequently are not operated continuously. Each unit requires about 40 gal. of water per minute.

The slip-mills were designed and patented by Mr. Macklind. Briefly, each mill consists of a tub of steel, 10 ft. in diameter and 3 ft. high, which is paved with carefully-cut blocks of Missouri granite. Over this pavement are dragged four granite blocks, roughly triangular, 36 in. on a side and 18 in. thick when new. There is a 3-in. bevel on the bottom along each edge. A heavy three-armed cap is clamped to the top. This cap has four holes about 6 in. apart along the leading side, and the dragging-arm is frequently changed from one to the other of these to change the direction of the drag and prevent corrugations in the bottom of the stone. Once in about 60 days the cap is loosened and the stone turned to present another leading-edge, which makes it wear evenly. At such times the bottom of the stone is usually roughened slightly. The stone should last for three years, but the wear on the bottom of the mill is about twice as rapid, and is, of course, greatest at the periphery. About once in two years the bottom will have to be raised or renewed.

Among the new features of this mill are an unusually heavy spider, and an arrangement of a weight and lever like a safety-valve, by which any fraction of the weight of the spider can

be transferred to the stones. This is adjusted to suit the different grades of barite. The weight-pin can also be lengthened to allow for the wear of the stone. The rest of the weight of the spider is carried upon a ball-bearing toe-piece raised above the water.

In the old-style mills trouble was caused by the tendency of the charge to form a self-supporting ring between the stones, which would then rub upon the bottom and wear more rapidly, and even at times break the gear or spider on account of the rapidly-increasing load. In this case the mill had to be stopped and the ring laboriously broken by hand. Mr. Macklind corrected this difficulty by providing the main-shaft with a ball-bearing collar, which can be raised 6 in. by a steam- or compressed-air piston. This also raises the stones, and the water soon causes the ring to collapse. It seems that this difficulty would be avoided if the distance between the stones were made greater by decreasing their number or increasing the diameter of the mills, according to the Mexican style. Mr. Macklind's scheme, however, has the incidental advantage that when the stones are raised the spider can be easily revolved by hand for inspection or repairs or for cleaning out the mill. In starting up it can also be brought to its normal speed of 20 rev. per min. by hand, so a simple positive clutch can be used on the gear.

When the mill is grinding properly, the stones tremble or chatter noticeably and the gearing runs without noise. When one of the stones begins to rub on the bottom the groaning of the gear should be immediately noticed by the attendant, who then raises the spider for a moment. The mills should have about 8 in. of barite in the bottom, and the attendant sounds them with a rod to determine the need of more feed, but of course soon learns how rapidly to shovel in the barite. At 20 rev. per min. each mill has a capacity of from 15 to 18 tons per 24 hr., but generally 12 or 13 tons is ground, and this output determines the capacity of the entire plant. About 15 h.p. is required for each mill.

The water flows through this mill at the rate of 100,000 gal. per 24 hr. This large stream of water may be necessary to provide for an increase in the capacity of the mill, but since more than 40 h.p. is required to raise the water 45 ft. to the

separating-tanks, it does not seem to be as economical as increasing the number of mills. The number of *spitzkasten* was increased in order that the floated barite should be fine enough. It is said that this fine barite was examined with a microscope, and the largest pieces were small enough to go through a 300-mesh sieve. The stream is divided equally among all the boxes, and enters through a fine screen to reduce the agitation. This screen also removes a few small pellets of clay and chips of wood. There is also a vertical screen in the middle of the box to prevent the formation of eddies.

The bleaching-tanks are lined with three lengths of 36-in. sewer-pipe without bells, and four special tiles form a cone-shaped bottom. They are made water-proof by an 8-lb. hard-lead covering on the outside. The hydrostatic pressure is resisted by oak staves and iron hoops outside the lead. The tanks are charged with 2 tons of settled barite, containing 25 per cent. of water, and a minimum of 240 lb. of 66 per cent. sulphuric acid is added. The acid is received in tank-cars, and costs a cent a pound. It is run into a large steel storage-tank, from which it is pumped by an acid-egg to an elevated lead-lined tank; thence it flows through lead pipes and valves to the bleaching-room on a level with the top of the bleaching-tanks. Here it is handled and measured in a small lead-lined tank-car. After the acid is added, steam is run into the bottom of the tank to agitate the mass and heat it to nearly the boiling-point. At first, lead-lined iron agitators were used, but they wore out very rapidly. The tile lining of the tanks protects the lead from all mechanical wear, and it soon becomes coated with lead sulphate.

It requires about 45 min. to heat the mixture, and the best quality of ore can be bleached in from 6 to 7 hr.; but since there is plenty of tank-capacity, the bleaching is always continued 12 hr. or more. If the bleaching is not finished in 12 hr., more acid is added; but it is unsafe to reduce the amount below 120 lb. per ton, on account of uncertainties as to the ore. No attempt is made to save the unused acid. To tell when the bleaching is complete, a 2-oz. bottle, full of barite, is compared as to color with a standard sample.

The bleached barite is drawn off through lead-covered plug-valves in the bottom. The handling of this hot, thick, heavy,

and corrosive material has proved very troublesome. It has been most successfully pumped to the washing-tanks by a very heavy acid-egg.

These washers, Fig. 5, designed by Mr. Macklind, consist of lead-lined iron tanks, 10 ft. in diameter, 19 ft. high, having a cone-shaped bottom. In a lead cup above the center of the tank the bleached mud is mixed with a strong jet of water, and then distributed by a flat lead cone to the circumference of the tank and discharged upon baffle-plates. In the center of the tank is

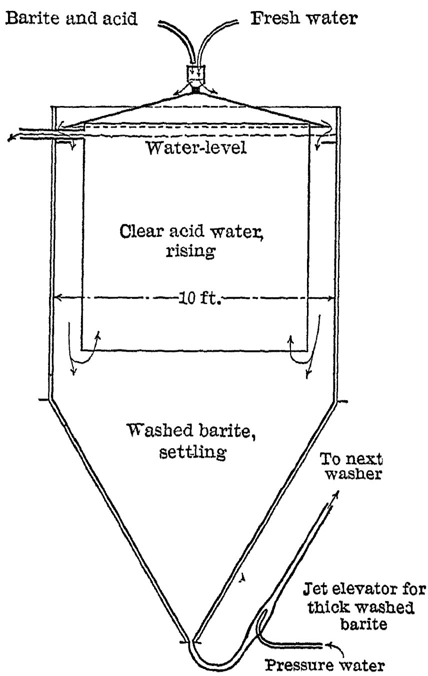


FIG. 5.—MACKLIND CONTINUOUS BARITE-WASHER.

a bottomless lead cylinder, 8 by 8 ft., inside of which the water rises so slowly that all the barite settles out, and only clear acid water is discharged to waste through a lead pipe at the surface.

From the bottom of the first washer the barite passes to a similar one 3 ft. lower. From this it goes to a third one still lower. Since, with this head, the heavy, settled barite would not flow from the bottom of one tank to the top of the next,

the current is maintained by a strong jet of wash-water directed upward in the discharge-pipe. A similar arrangement might be serviceable in elevating the thick barite from the bleaching-tanks.

All the acid should not be removed or the barite will immediately turn a light yellow; the end-point used to be determined by a slight acid taste, but the washing is now continued until there is no taste, but the iron sulphate remaining in solution will give a slight blue color with potassium ferrocyanide, thrown upon the last spreading-cone. When this does not show, the quantity of wash-water is reduced. The reason for the change of color when no acid is present is not understood. This acid prevents the use of floated barite for weighting rubber and paper, but for paint it is not objectionable. If the settled barite mud at the bottom of the washer has 50 per cent. of water, this leaves about $\frac{1}{8}$ lb. of acid to the ton of barite; if it has 100 per cent., which is more likely, 1 lb. to the ton will be left.

The wash-water is filtered through six sand-filters, 36 by 60 in., and is obtained from a small creek by a separate pump. The general mill-supply is filtered only when very muddy.

The essential part of the Macklind drier is a cylinder of wrought-iron pipe, 36 in. in diameter and 8 ft. long, revolving one-and-a-half times per minute, and heated inside by live steam. Upon this cylinder the barite is dropped by a plate shaken 200 times per minute by a cam. The thick, settled barite mud is fed upon the shaker-plate in fine streams 4 in. apart through pin-valves from the feed-trough, which contains an agitator to prevent packing. The drops of barite falling upon the hot drum are soon dried and are scraped off by a plate after nearly a full revolution. This scraper leaves a thin layer of strongly-adhering barite, which prevents discoloration from the iron. The live steam enters through a stuffing-box at one end, and the condensed water is drawn off through a siphon-pipe in the other trunnion and circulates through coils along the side of the housing, and dries whatever barite is spattered off by the boiling pellets.

To prevent the accumulation of air in the cylinder, some steam is drawn off through a straight pipe entering through the siphon and carried to the bleaching-tanks.

The finished barite is tested for color by putting a spatula-

full upon a glass plate beside a standard sample, smoothing it down beneath a piece of paper, and comparing the color. It is then moistened with turpentine and again compared. In this way it shows as white as a standard sample of white-lead, which had a slight buff color, but not as white as good zinc oxide, which is blue in comparison. The product is tested for grit by rubbing it with a spatula upon soft glazed paper, which should not be scratched. Barite which contains flint will be apt to show more grit when floated, since the equal-falling grain of quartz is so much larger than the heavy barite. Barite ground between burr-stones gets grit from the mills. In the wet-process mill, in which the barite is ground before bleaching, hard iron or steel could be used in the mill and the pigment would be entirely free from grit if the barite contained no quartz.

Barite, on account of its prismatic cleavage, forms a "sliver" pigment rather than a granular one. These slivers mat together in the paint, and since barite is slightly harder than the other pigments, it forms a more durable paint. It is also entirely unaffected by weather, and does not dissolve like zinc, or blacken with sulphur like lead.

White-lead is said to saponify linseed oil. At any rate, after a few years, a pure white-lead paint will chalk and rub off, leaving the surface in good shape for a second painting. So far as known, zinc is not used alone, but if the paint contains too much barite it peels off and a second coat of paint will not stick. The mixture of white-lead and barite pigments is therefore better than either one alone. Some of the peeling may be due to inferior oil.

When barite is sufficiently ground it will spread well. The chief objection to it is its translucency and lack of covering-power. This deficiency is best corrected by zinc oxide, which has the greatest covering-power. Some of the paint-men therefore say that an equal proportion of the three pigments makes the best paint. The unfortunate feature is that usually good barite costs less than a cent a pound, while the other pigments cost 6 or 7 cents, so that much of the prepared paints contain too great a proportion of barite, up to 60 per cent.

Very little barite is now used in making fine book-paper, since the weight of a book is no longer considered an advan-

tage, and fibrous talc is used instead. Its use in rubber goods seems to be a trade secret.

For making barium salts a very inferior barite can be used, but most of this chemical manufacturing work is done in Europe. There would be a big demand for barium oxide for treating boiler-water containing much calcium and magnesium sulphates, but it is not yet supplied cheaply enough in the United States.

I desire to express my cordial thanks for information regarding Washington county barite to W. R. Macklind, of Mineral Point; James Long and James W. Richeson, of Potosi; Frank Long, of Cadet; and Robert McClay, of Bliss, Mo.

Conditions and Costs of Mining at the Braden Copper-Mines, Chile.

BY WILLIAM BRADEN, NEW YORK, N. Y.

(Spokane Meeting, September, 1909.)

THIS paper is presented in the hope that it will be instructive in view of the future large expansion of the mining industry in the west-coast countries of South America.

There is a more or less general impression that the Spanish-American workman is inferior to the American, but after some years of experience and observation I doubt the correctness of this view. Taking into consideration all the elements which make for efficiency of labor, it has been found, particularly in Chile, that under proper organization native labor yields as much, man for man, and more, dollar for dollar, than in the Western United States.

If a manager is willing to accept unreservedly the *costumbres del pais*, without combating intelligently and patiently the ones tending to inefficiency, he should expect no better results than he deserves. He must conscientiously insist that a "square deal" be given and exacted; that liquor be excluded from camp as far as possible; that comfortable quarters and other uplifting elements of life be provided; and that past and existing methods and customs shall not be over-ridden rough-

shod. He should insist upon the gradual and reasonable adjustment of these conditions to the exigencies of the work; and (by no means the least difficult of his tasks) he should select the most competent men to direct and teach the natives patiently in the several departments, and by their own example encourage self-respect and decency.

While the Braden mine is by no means as yet operated on a large scale, according to modern rating, it will nevertheless be interesting to note what has actually been accomplished there. An illustrative description of the mines and mill of the Braden Copper Co. has already been published.[1] However, in order to lend value to the figures given, the general conditions will be explained.

The copper-deposit is a zone of mineralized, fractured, and brecciated diorite, 150 ft. wide, lying under a hanging-wall of brecciated tuff, which has a dip of approximately 65°. The mine is dry, and little or no drainage of any kind is necessary. The system of mining may be described as overhead mining on broken ore, in a series of transverse stopes, 10 m. wide, with intervening pillars, 7 m. wide, which will be extracted ultimately by the caving system. The ore is moderately hard, and the roofs of stopes, 10 m. wide by 50 m. long, stand perfectly well without timbering. The only timbering in the mine is for the framing of the ore-gates for extracting ore from the stopes.

The air-drills, used to the fullest extent, are handled carefully and intelligently by the Chilean miners, and the results have been so favorable that eventually hand-drilling will be almost entirely supplanted.

Rack-a-rock is the principal explosive used; but dynamite and black mining-powder (the latter manufactured in Chile) are also used to a limited extent.

The ore is trammed out of the mine through a main adit in 1-ton cars by means of a three-phase electric haulage-system (220 volts). It is then carried to the mill, 2,630 m. distant and at 550 m. lower elevation, by two Riblet-system aërial tramways.

Water for the hydro-electric power is supplied through a flume, 30 in. wide by 20 in. high, and about 6,000 ft. long, and

[1] *Engineering and Mining Journal*, vol. lxxxiv., No. 23, pp. 1059–1062 (Dec. 7, 1907).

a pipe-line from 22 to 18 in. in diameter, which leads to three 42-in. Doble water-wheels, two under 820-ft. head, and one under 840-ft. head, direct coupled to a 200-kw., 2,200-volt alternating-current generator. Current is transmitted about a mile and a half to one motor running under the same voltage, belted to drive an air-compressor at the mine, and to various other motors (current transformed to 220 volts) for machine-, blacksmith-, and pattern-shops, and (110 volts) electric sample-driers and electric-light system.

The following is an extract from the operating-report for November, 1908. Costs are given in United States currency, and weights in dry tons of 2,000 pounds.

The total cost of breaking 16,185 tons, including superintendence and general charges, was 41 cents per ton.

Upon the basis of 7,304 tons extracted and milled, the costs per ton were distributed as follows:

Ore-breaking,	$0.31
General mine-expense,	0.06
Development,	0.12
Underground tramming,	0.02
Aërial tramming,	0.06
Milling: Operation,	0.27
Repairs (labor),	0.01
Repairs (materials),	0.09
General mill-expense,	0.05
Power,	0.01
Sampling and assaying,	0.05
General expense,	0.23
Taxes, insurance, and interest,	0.04
	$1.32

The ore-breaking account includes much cutting-out for ore-pockets, sub-cross-cuts, raises, and other work of a preparatory nature. More than one-half of the labor was performed by contract. In the stopes, contracts were let on the basis of the number of feet of hole drilled. The actual cost of the labor for one month was:

164 man-days (9 hr.), with 2¾-in. "New Ingersoll" air-drills, drilled 5,082 ft. at a cost of 2.025 cents per foot. Average amount drilled per man-day, 31 ft. 1,008 man-days, by single hand-work, drilled 13,855 ft. at a cost of 8.2 cents per foot. Average amount drilled per man per day, 13.8 feet.

Including all labor, both contractors and day-pay men

employed for mining, there were 12 tons of ore broken per man-day.

Since the above date, a system of contracting for ore-breaking by measurement of the ore broken has resulted in a considerable reduction of the cost for that item in the foregoing statement.

Development was in the nature of drifts, cross-cuts, and upraises, with an average section of 4.53 sq. m., and, including all labor, supplies, explosives, mucking and tramming, the cost was $3.54 per foot driven, or 78 cents per cubic meter.

The amount of driving was 379 lin. ft., with 876.75 man-days, making 13 lin. ft. per month per man, or 0.433 ft. per day, and 1.96 cubic meters.

To give a concrete idea of what is considered to be good work, a contract was let for driving a main adit of a section of 6 sq. m., at a total cost of $3.15 per running foot, exclusive of drill-sharpening and repairs; 132.25 ft. were driven in 30 days.

Payment of all labor accounts is made three times in each year, on Jan. 1, May 1, and Sept. 1. The workmen are permitted, however, to draw up to 80 per cent. of their balances at any given time—a custom of the country which leads to constancy of work and gives entire satisfaction. Wherever possible, a bonus is paid to encourage good steady work.

The foregoing data indicate most conclusively the efficiency of Chilean labor.

Preparing and Recording Samples for Use in Technical Assay-Laboratories.

BY LOUIS D. HUNTOON,* NEW HAVEN, CONN.

(Spokane Meeting, September, 1909.)

AFTER the completion, in 1905, of the Hammond Mining and Metallurgical Laboratory of the Sheffield Scientific School, Yale University, it became necessary to secure and assay a large assortment of ore-samples, and to arrange the results in such a manner as to admit of use with the least amount of work on the part of the instructor.

At first, pulp-samples, together with their respective assays, were secured from the mines, smelters, and public assayers, but this practice did not prove satisfactory. It was not only difficult to secure a sufficiently large supply of material, but it was also impossible to secure the proper assortment required for instructing students. It involved considerable work for the donors to select the samples and furnish a record of the assays, and also for the instructors to examine each pulp and determine its character. Some of the samples did not contain a sufficient quantity of pulp for the student to check his work in case his first assay did not correspond to the record.

In order to increase the number of samples and to vary the character of the material, pulp-samples were later mixed with barren rock and with sulphides, but this method also was unsatisfactory and has been discontinued.

At present, small lots of ore are secured by gift or purchase from engineers, public assayers, and the managers of mines, mills, and smelters. These lots weigh from 10 to 100 lb. each, and are rarely finer than 0.25 in. in size. The larger samples permit of a careful study by the instructor in order to determine the best method of assaying to be followed.

In universities in which assaying is taught the record of the samples should fulfill the following requirements:

---

* Professor of Mining and Metallurgy, Sheffield Scientific School, Yale University.

1. Ease of record, avoiding all unnecessary duplication of work, and, so far as practicable, the possibility of error from duplicating numbers or in recording the assay.

2. Filing of assay-records properly, especially if the card containing the record has been taken from the file for comparison.

3. Facility in securing the character of sample desired.

4. No duplicating of numbers for the same sample.

5. The number of the last sample recorded should be easy of reference.

6. The number on the sample-sack should not in any way indicate to the student the character of the pulp contained.

7. Finding promptly the assay-record of a sample by having only the number on the sack.

8. The total number of different samples of any character should always be known, so as to replenish the stock when necessary.

The Dewey decimal system of record was adopted so as to indicate: 1, whether the sample is an ore or a furnace-product; 2, the character of the gangue; 3, for what metals the sample is to be assayed; and 4, the method of assaying to be used by the student. This system has been in use at the Hammond Laboratory for four years, and with a few slight changes has proved entirely satisfactory.

The records of the assays are entered on cards, 5 by 8 in. in size, indexed as shown in Fig. 1. Numbers in the left-hand column on the key are placed to the left of the decimal point on the assay-cards and indicate whether the sample is an ore or a furnace-product, and also for what metals the sample is to be assayed. Thus, 23 indicates gold and silver; 13, lead and silver; 83, a scale-ore of silver, such as ores from Cobalt; 4123, a lead-bullion containing gold and silver; and 913, a special furnace-product containing lead and silver.

The second set of numbers are placed to the right of the decimal point and indicate the character of the gangue and the method to be employed in assaying; thus, .13 indicates the gangue to be a basic oxide and may be either limestone or an oxide of one of the base metals; .14, the ore has a siliceous gangue which requires a basic charge with the addition of reducing flux; .5, the ore is a sulphide, 1 g. of which will reduce

more than 2.5 g. of lead, and should be run by the crucible-method with the addition of nails; .6, the ore is a sulphide, 1 g. of which will reduce between 1.3 and 2.5 g. of lead, and can be run by the crucible-method with the addition of niter; .7, the scorification-assay will give higher results, otherwise it is an impure ore giving a matte or speiss by the crucible-method. The decimals .57 and .67 indicate that the gold is

KEY.

ORES.		GANGUE.		
1	Lead.	.1	Oxide.	
2	Gold.	.2	Sulphide.	
3	Silver.	.3	Basic.	
PRODUCTS.		.4	Acid.	
4	Bullion.	**METHOD.**		
5	Speiss.	.5	Nail.	
6	Matte.	.6	Niter.	
7	Cu bar.	.7	Scorification.	
SPECIAL.		.8		
8	Scale ore.	.57	Nail for gold ; scorification for silver.	
9	Special.	.67	Niter for gold ; scorification for silver.	
10				
11	To be assayed.			

Fig. 1.—Key to Classification of Assays.

to be determined by a crucible charge, using the nail- or niter-method respectively, and the silver by scorification.

The index- and assay-cards are contained in ordinary filing-cases. One file contains the records of the common ores and another the records of the furnace-products, and ores and products requiring special treatment. The first index-card to the left, containing only the number to the left of the decimal, is of one color, and the remainder of the index-cards, indicating both the content and the character of the sample, are of another color. Cards showing the same character of sample and method of

assaying are placed in the same relative position for each set of ores, an arrangement which greatly assists in the finding of a desired sample.

The assay-card, shown in Fig. 2, contains a complete record of the sample, entered from the instructor's assay-certificate, shown in Fig. 3. The series number in the upper left-hand

SERIES 23.67	Au. 2.31	Pb. 5 % ±	FeS_2 25 % ±		Samples Nos.
	Ag. 36.20	Cu. 2 % ±			1492 to 1507

CHARACTER *Screen Test—10–20—Calumet.* Assayer *Jones & Brown.*

1492	1493	1494	1495	1496	1497	1498	1499
Checked by							
Jones.			*Smith.*	*Asher.*		*White.*	
$\frac{2.30}{36.27}$			$\frac{2.24}{35.80}$	$\frac{2.28}{36.10}$		$\frac{2.36}{35.70}$	
✓			✓	✓		✓	

1500	1501	1502	1503	1504	1505	1506	1507
Checked by							
Brown.		*Jackson.*	*Moore.*				*Macy.*
$\frac{2.32}{36.13}$		$\frac{1.97}{30.10}$	$\frac{2.35}{35.80}$				$\frac{2.26}{34.90}$
		✓	✓				✓

6

FIG. 2.—RECORD OF ASSAY.

corner of the assay-card, 23.67, indicates that the ore contains gold and silver, the gold to be determined by the crucible-method with the addition of niter, and the silver by the scorification-method. The number in the lower right-hand corner indicates that the card is the sixth in position under the index-number, 23.67. At the top of the card are entered also the results of the fire-assay, the estimated percentage of base metals determined by vanning, any note of special interest under "char-

LABORATORY ASSAY.

Character of Gangue.	.13—Acid .14—Basic.... ....Fe........, Ca........, .2—Sulphide Fe 25%.... Pb 5%..... Cu 2% ±... As Sb........,
Series 23.67.	PRELIMINARY ASSAY. 1 Gm. Ore { Oxidizes...... Reduces........2........Gms. Pb.
Assay Charge.	Ore........½........½........ .2........A.T. PbO.......20........60........Gms. Pb..........2 *nails*........60........ " Niter........4........ " SiO_2........*Bx Gl* 2........ " Cover......*Bx Gl*........*Bx Gl*........ "

Time in furnace...50...........60..........50

Slag: Color...*Black*....... *Green*......... Character...*Basic*....... *Glassy*........

Button: weight...18.........20.........22 Character...*soft*........

ASSAY AND METHOD.

Litharge .1		Nails .25			Niter .26		Scorification .27	
2. Au.	3. Ag.	1. Pb.	2. Au.	3. Ag.	2. Au.	3. Ag.	2. Au.	3. Ag.
			1.12	16.30	1.14	17.50	.42	7.20
			1.10	16.50	1.16	17.20	.44	7.30
			2.22	32.80	2.30	34.70	.41	7.24
							1.27	21.74
							5	5
23.67							3) 6.35	108.70
							2.12	36.27

No. 1492......Date...3/21/07........Assayer......*Jones*........

FIG. 3.—INSTRUCTOR'S ASSAY-CERTIFICATE.

acter," and the names of the assayers. The cards are ruled to contain 8 samples, but in case there are 16 duplicate samples these are usually recorded on the same card. The first samples recorded on each line are assayed by different instructors. The names of the students are entered under the number of the

sample assigned to them, and when the samples are returned the series-number and students' assays are entered on the pulp-sack and later checked on the assay-card as having been returned. If the sack returned contains too little pulp to be of value, it is destroyed and the number on the assay-card crossed out. If the returned sack contains sufficient pulp to be of value the number is checked and the sample filed. When all of the samples recorded on one card have been given out and are also returned, the pulps are re-mixed and divided into new samples, and new numbers are given to each one, the record being entered on a new card. One sample of each lot of mixed pulps is assayed by an instructor so as to avoid the possibility of error arising from a student placing the pulp in a wrong sack.

Duplicate samples are given out under different numbers, and rarely does any one class of students receive more than two samples from the same card. The samples, as mixed, are numbered consecutively, regardless of the contents. Card 6, series 23.67 (Fig. 2), records pulps 1492 to 1507; the following card, 7, of the same series, may record pulps 6001 to 6016. Pulp 1507 is in series 23.67 and the following number, 1508, may be in series 1.5. By this method the number on the sack does not indicate to the student the character of the sample or the method to be employed in assaying.

In case it is desirable to find the assay of any special number, the series of which is not recorded on the sack, reference is made to the location-card, Fig. 4, which contains the numbers of the samples and the series to which they belong.

For instruction-work it is desirable to have as large an assortment of samples as possible with different assays. At present there are recorded and ready for use in the Hammond laboratory about 7,000 samples. The annual consumption amounts to about 2,000 samples, and the new ones added to the list each year amount to about 3,000, which gives a gain of about 1,000 samples per year. It is hoped in the near future to have on record 25,000 samples, which should afford a sufficiently large assortment of assays requiring different methods of treatment. In order to obtain this variety, products are used from the testing of ores, from screen-tests of 25-lb. lots, from mixing 5-lb. lots of comparatively coarse gold-ore with different quantities of pyrite, and silver-ore with different amounts

of galena. The samples thus obtained are pulverized in jar-mills and passed through an 80-mesh screen. For 16 gold-pulp samples, weighing about 125 g. or 4 A. T. each, 2,000 g. of pulp is taken, thoroughly mixed on a rubber sheet, and riffled into two lots, marked A and B. These lots are separately mixed and riffled into eight samples each, mixing thoroughly between each riffling, and placed in numbered pulp-sample bags. The

7233
7528

7233		~~7285~~	3.5	7303	2.6	7363		7427	
to	2.14	~~7286~~	3.5	7311		7364	123.57	to	2.14
7248		7287	32.7			7365		7443	
7249		7288		~~7312~~		7366	23.5	7444	
to	2.5	7289	2.14	to	3.7			to	2.14
7264		7290		~~7328~~				7460	
7265	1.5	7291	2.5	7329		7367		7461	
7266	1.5	7292		to	623.	to	13.7	to	3.7
7267	2.7	7293		7336		7375		7469	
7268	23.14	7294	23.5	7337		7376		~~7470~~	
		7295			23.67	to	3.5	to	2.7
		7296	13.5	7353		7392		~~7478~~	
7269		7297		7354		~~7393~~		7479	
to	123.14	7298	62		123.13	to	2.6	to	13.7
7275		7299	62	7362		~~7417~~		7495	
~~7276~~		7300	83			7418		7496	
	23.57	7301	83			to	1.5	to	2.14
~~7284~~		7302				7426		7528	

FIG. 4.—INDEX-CARD FOR LOCATING SERIES-NUMBER OF SAMPLES.

numbers of lot A are entered on the top line and the numbers of lot B on the lower line of the assay-record cards. The cards are then filed under No. 11, to be assayed, and the samples placed in stock. The first recorded samples of each lot assayed by different instructors must check within a commercial limit

before the record is accepted. In case the assays do not check, which is rarely the case, the samples are re-mixed and re-assayed.

The weight of the pulp in the sacks varies slightly, depending on the character of ore. Ores requiring charges of 0.5 A. T. weigh about 120 g.; low-grade ores requiring charges of 1 A. T. about 220 g.; and scorification-ores about 100 g. Each sack contains sufficient ore for preliminary vanning and testing, and for two sets of assays if necessary. The student reports his results, and in case his results do not check, he does not consume unnecessary time in re-assaying without personal instruction.

On the assay-certificate, Fig. 3, used by the instructor in this work, the number of the sample is entered on the lower left-hand corner, followed by the date of the assay and the name of the assayer. The "character of the gangue" is either checked before pulverizing or determined by vanning-tests, preferably the latter, as the student determines the character of the sample by this method. The preliminary assay is made, if necessary, and the oxidizing- or reducing-power entered on the certificate. The weight of the charge used and the notes taken on the furnace-work are next entered under the heading "Assay and Method."

The above information gives the series number, which is next entered, thus completing the certificate for file and entry on the assay-card. These certificates are kept on file for future reference in case an assay is disputed.

In laying out a term's work for a class it is necessary to select a large number of different samples having the same general character and method of assaying. The entire assortment of samples of gold-ore requiring a basic charge will be found under 2.14, 23.14, and 123.14, the number .14 remaining in the same relative position in the file. In like manner gold-ores to be run by the niter method will be found under 2.6, 23.6, 23.67, 123.6, and 123.67.

Although at first sight the above system may appear to be complicated and to call for a large amount of work, in practice it has worked out very satisfactorily. The system has been evolved during the past four years, until at present there is but one entry from the assay-certificate to be made on the record-card. Two duplicate assays, as a rule, suffice for recording the assays of 16 samples.

Glass Mine-Models.

BY EDMUND D. NORTH, TONOPAH, NEV.

(Spokane Meeting, September, 1909.)

In making a glass model of mine-workings, each mine will present some little individualities, to meet which will call for the exercise of special ingenuity. Having made several models, I offer the following details of the model of the Montana-Tonopah mine-workings, believing that with small changes for different conditions the method described can be successfully used in making a glass model of the workings of any mine.

The model, Fig. 1, has the top, one end, and the front removed. The case is made of kiln-dried pine, painted with several coats of dead-white paint. It is made to take sheets of glass 24 by 42 in. in size, and is 18 in. high, the workings shown being on a scale of 1 in. = 40 ft., or 4800 : 1 scale.

The base is made of four pieces of 2-in. material, well pinned and glued together. The back, of material 1 in. thick, is secured rigidly to the base by screws and countersinking.

The four posts, 1.5 in. square, are so cut that the front and two ends of the model fit in flush with the outside edges of the posts. The top rails, 1 in. square, are fastened rigidly to the front and back posts by means of small iron braces.

The ends, front, and top of the case are of 0.5-in. material; the ends and front are held in place at the bottom by pegs fitting into the base; at the top the ends are held by buttons and the front by cupboard-hooks on the inside. Hinged hasps on the cover, fitting over staples in the end-pieces, make it easy to lock the entire case.

Before the back is fastened to the base, it is cut with horizontal and vertical grooves to receive the horizontal and vertical sheets of glass, shown clearly in Figs. 1 and 2. The horizontal grooves, spaced to scale, represent the vertical distances between the levels at the shaft. The vertical grooves are placed wherever cross-sections of the workings are desired.

The posts have horizontal grooves to correspond with those in the back. Strips of galvanized iron, 0.5 in. wide, and bent throughout their entire length so as to have two 0.25-in. faces at right angles, are fastened by small screws to the front and back posts, the upper surface of the iron being flush with the bottom of the horizontal grooves. This construction is very rigid, and the view through the ends of the model is not materially obstructed. Without these iron strips the horizontal sheets of glass do not have the proper end-support, and tend to crack at the corners.

Eight 1-c-p. electric lights (frosted), placed in the base, make it possible to see clearly every part of the model.

The horizontal sheets, of ordinary double-thick glass, selected as free from bubbles and imperfections as possible, are 24 by 42 in. in size. The cross-sections are made on ordinary single-thick glass, cut 24 in. long, and in widths to fit closely between the respective horizontal sheets. These vertical strips are used, in places, from the base of the model to the top, even when not used for sections, because of the support they offer to the horizontal sheets.

Fig. 3 shows the means devised to keep the vertical sheets of glass from moving laterally on the horizontal sheets at the front of the model. These little clips are made of brass spring-wire, and as the thickness of the glass varies, they can be made to hold firmly anywhere. They offer very little obstruction to the view.

As the first mine-level is 400 ft. below the shaft-collar, because of a capping of about 370 ft. of barren andesite, the model shown in Figs. 2 and 3 starts at about 350 ft. below the collar. The lowest horizontal sheet in the model has the coordinates and claim-lines on it, this one sheet serving to orient the observer for all levels.

In case the model represents the mine up to the surface, a top sheet, showing the topography and surface-improvements, is often desirable.

If ground glass be used in place of clear glass for the bottom sheet, it will do away with reflection from the lamps, which may be noticed at some angles if clear glass is used.

A model as above described can be opened in a moment, and a clear view can be obtained—from either end through the

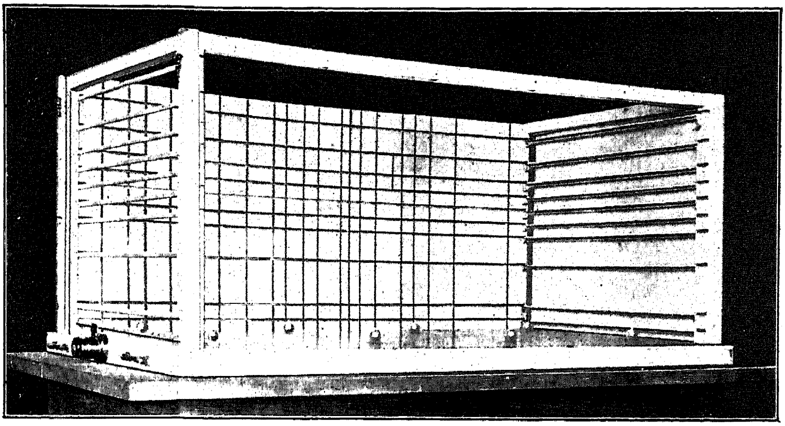


FIG. 1.—CASE WITH TOP, FRONT, AND ONE END REMOVED.

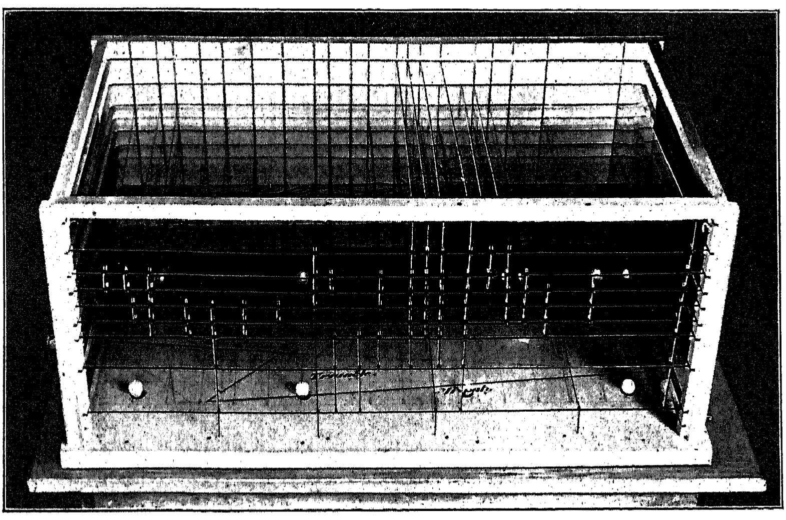


FIG. 2.—MODEL OF MONTANA-TONOPAH MINE-WORKINGS.

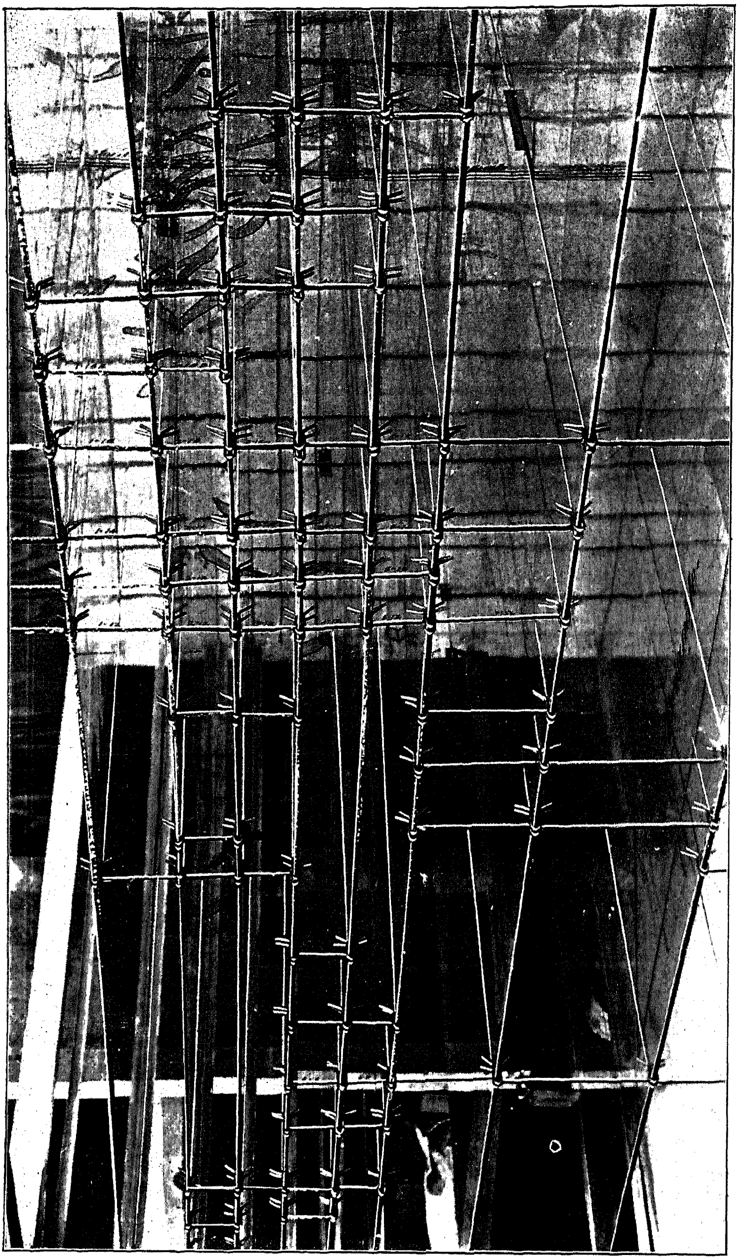


FIG. 3.—VIEW SHOWING WIRE CLIPS TO PREVENT LATERAL MOVEMENT OF THE VERTICAL GLASS SHEETS.

sections, from the top looking at the plans, or through the top at an angle, observing the plans and sections at once.

All the work is first platted on paper, and the glass plate is used the same as tracing-cloth would be.

Winsor & Newton oil-colors are used for drawing on the glass; an equal mixture of turpentine and white japan is made up in stock, and this is thoroughly mixed with the oil-color in a small dish as needed. Different colors require different amounts of this mixture added to them, the proper mixture for each color being obtained when it will flow freely from a ruling-pen to the glass and still be thick enough to stand permanently on the glass and not spread.

Free-hand lines and lettering can best be made with a ruling-pen, since on a glass surface this pen makes even lines in all directions. About 24 hr. is required for the mixture to dry; the levels can then be filled with their respective colors with a brush. It is best to have the outlines of the same color as that with which they are to be filled.

All sections being numbered, it is a simple matter to take the model apart and to put it together again, so that the work can be added to regularly and the model posted to date; new grooves can be cut in the back and new sections added wherever desired.

The sections in this model are north and south, and in any model it is well so to orient the work that the desired sections will be at right angles to the back of the model. All workings which are at or within 10 ft. of a section are put on the section in full black; if desirable to show anything further away than 10 ft., it is dotted, showing it to be projected. By using care in choosing the sections, very little projecting will be found necessary.

The colors used for the levels are the same as those used on the working-map of the property—namely: first level, carmine; second level, blue; third level, yellow; fourth level, green; fifth level, purple; and intermediate levels, gray.

In sections, all work in ore is outlined in red; if desirable, the geology can be placed on each level and on each section by cross-hatching the different formations in different colors, which will show each formation plainly without obstructing the view.

An Adjustable Pyrometer-Stand.

BY L. W. BAHNEY,* PALO ALTO, CAL.

(Spokane Meeting, September, 1909.)

FREQUENTLY in using a thermo-electric pyrometer for measuring the temperature of a furnace, a hole is drilled at the back or side of the furnace, through which is introduced the tube containing the thermo-couple. At times the couple is left almost where it drops, for the reason that it soon becomes too hot to be handled easily, and the space at the back or side of the furnace may be so small and uncomfortably hot that an easy and accurate adjustment is nearly impossible. In order to overcome this clumsy and unscientific method of using the thermo-electric pyrometer, I designed an adjustable stand, to hold the clay or quartz tube inclosing the platinum and platinum-rhodium leads that constitute the couple of the Le Chatelier thermo-electric pyrometer. This stand also allows an easy and rapid adjustment of position to varying heights and angles, as may be desired for special reading.

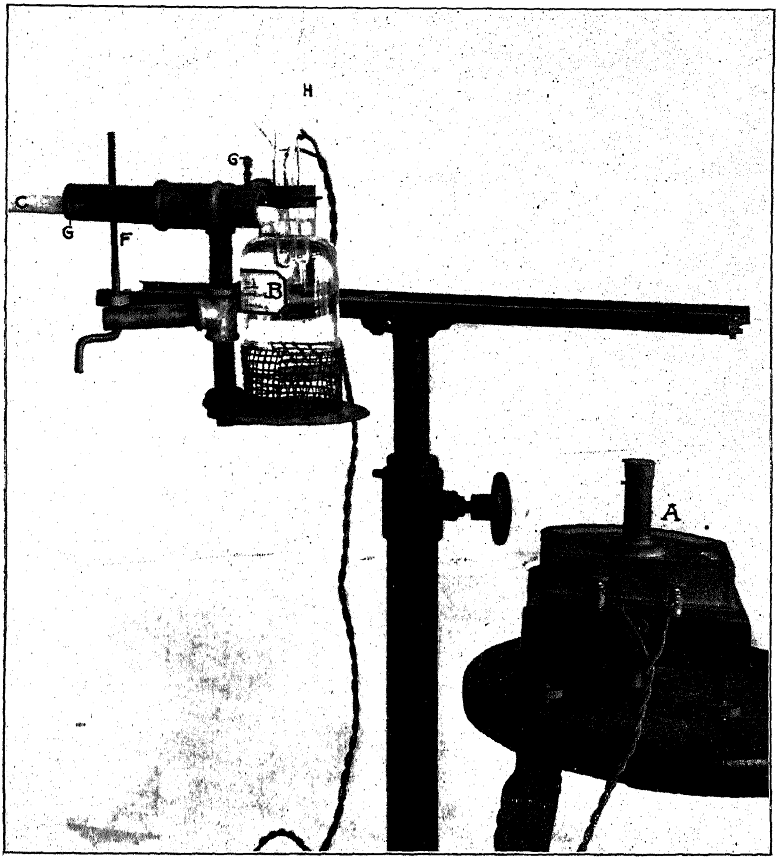
This stand is illustrated in Fig. 1. The couple in the tube might be of better service if the tube were held a little higher above the charge in a roasting-dish, changing the height and angle if desired.

The stand, made of 1.5-in. water-pipe and mounted on casters, is heavy enough so it cannot be tipped over easily. It is adjusted to any position, and has a universal clamp which holds the clay or quartz protecting-tube.

To avoid disturbing the wires of the couple at the cold junction each time the tube is moved, the cooling-bottle sets in a wire basket so placed that it moves forward or back with the couple.

To overcome the necessity of leveling and adjusting a galvanometer at each furnace, a switch-board is used to connect

* Instructor in Metallurgy, Stanford University.



A. Siemens-Halske galvanometer.
B. Cold junction filled with water, containing small U-tubes filled with mercury, by which the contact is made with the couple-wires and the lead to the galvanometer.
C. Quartz protecting-tube, 0.5 in. bore.
F. Fine-adjustment screw.
G. Thumb-screws for fastening the protecting-tube.
H. Thermometer.

FIG. 1.—VIEW OF ADJUSTABLE PYROMETER-STAND.

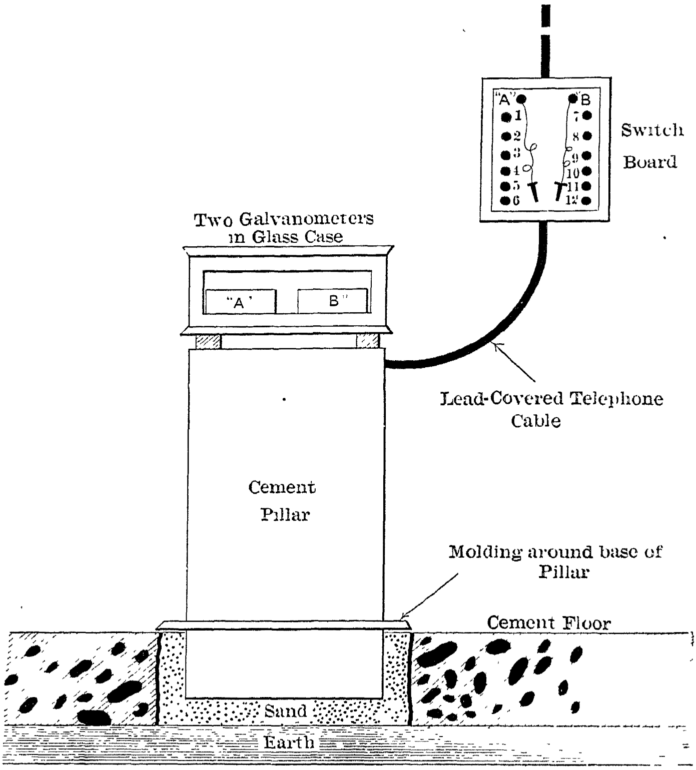


FIG. 2.—DETAILS OF CONSTRUCTION OF CONCRETE PILLAR.

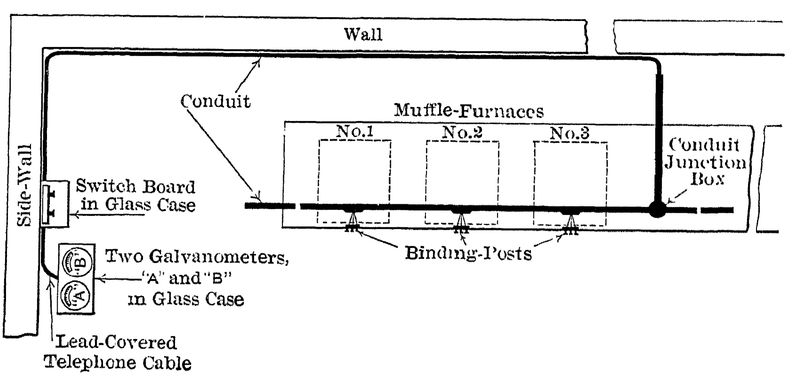


FIG. 3.—PLAN SHOWING ARRANGEMENT OF FURNACES, SWITCH-BOARD, AND PILLAR FOR GALVANOMETER.

the pyrometer with the galvanometer by means of three binding-posts placed at each furnace.

The furnaces are numbered from 1 to 12, and to facilitate the work two galvanometers are used, *A* and *B* in Figs. 2 and 3.

On a block in front of each of the furnaces, Fig. 3, are binding-posts *A* and *B*, to correspond to the galvanometers, while the central post is numbered according to the furnace. Thus, the binding-post block of furnace No. 3 would be marked *A 3 B*.

The connection between the binding-posts and the switch-board is by means of a telephone cable cased in a lead tube. Beneath the furnace this tube is inclosed in a conduit for protection.

The switch-board is of marble, having a plug for each of the galvanometers, *A* and *B*, and 12 holes, each bearing the number of a furnace, so that any individual furnace may be connected with either galvanometer, or with both.

A perfect adjustment of the galvanometers is obtained by placing them in a dust-proof case mounted on a concrete pillar, 1 by 2 ft. by 4 ft. high, resting in sand about 1 ft. below the floor, as shown in Fig. 3.

The wooden base, which merely covers the hole in the floor at the front, back, and sides, is fastened to the pillar, so that any motion of the floor is not communicated to the galvanometers.

In calibrating the couples, the readings of the galvanometer are taken through the cable and switch-board, so that similar conditions will obtain when in use.

For general work, the Battersea clay tubes made by the Morgan Crucible Co., England, are the best, as they are not so easily corroded and broken as are the quartz tubes.

The Cyaniding of Silver-Ores in Mexico.

BY ALBERT F. J. BORDEAUX, THONON-LES-BAINS, FRANCE.

(Spokane Meeting, September, 1909.)

THIS paper briefly describes the general outline of cyaniding silver-ores in Mexico, with special reference to personal experiments made in the Temascaltepec district.

The most important papers on the subject deal with cyaniding gold-silver ores, the gold predominating in value, so that the treatment is nearly the same as for gold-ores, the losses of silver being considered of little importance. I describe here the treatment of silver-gold ores containing very little gold, which is more general in Mexico than elsewhere.

The cyanide treatment, which was so successfully applied to gold-ores, did not succeed immediately with silver-ores on account of their varying composition. Gold occurs in nature usually in the native state, while silver occurs generally in combination with sulphur (argentite), and various combinations of sulphur, antimony, and arsenic.

In former times there was a complete difference in the treatment of gold-ores and silver-ores. Amalgamation, which was successful with gold-ores, was generally inadequate with silver-ores. The usual treatment in Mexico for silver was the *patio* process, the theory of which was satisfactorily explained only quite recently by Mr. Ortega and others.

GENERAL OUTLINE OF THE CYANIDE TREATMENT.

The general tendency of the present practice is to slime the silver-ore as much as possible, and in order to obtain a higher efficiency, with less loss of cyanide, and greater speed, the following lines are observed:

1. Crushing by stamps with 30- or 40-mesh screens, then crushing by Huntington or Chilean mills, through 60- or 80-mesh screens.

2. Concentration upon vanners.

3. Separation of sands and slimes with *spitzkasten* or cones.

4. Sliming in tube-mills to 100- or 120-mesh screens, or even to 150-mesh, then separation again by cones for either treating the sands separately or recrushing them in tube-mills for all-sliming.

5. Treatment of the sands, if isolated, in filter-tanks.

The slimes can be treated only by decantation; the new Butters slime-filter, as used in Nevada, does not seem to be successful with silver-ores. An "all-sliming" plant requires less outlay of capital, and less daily expense, because it does away with a second, and sometimes a third, treatment required for the sands.

OPERATIONS.

Crushing.—Crushing in solution can be done only with gold-ores, as, for instance, at El Oro, Dos Estrellas, etc. With silver-ores containing very little gold, crushing in water is imperative, as there would be some heavy losses of solution in running the pulp to the concentrating-tables, pumps, tube-mills, etc. The problem is to determine the best proportion of slimes and sands to be obtained, considering different factors, such as the cyanide-expense, the length and simplicity of treatment, and the highest extraction. A comparison of the various systems used in Mexico, leads to the conclusion that the evidence tends to favor the all-sliming treatment.

The best mill for sliming is the tube-mill. In order to use it at its full capacity, the general practice is to feed sands alone, which have first been separated from the slimes in a series of separating-cones.

Cone-Separators.—These cones are used instead of *spitzkasten*, because of easier construction and more rapid action. The separators, made of galvanized iron, are open in the lower part, and have an outer rim on the upper part. The inner part is a cylinder of the same sheet-iron, reaching nearly to the bottom. The upper diameter of the cone varies from 1 to 1.3 m., and the height is from 1 to 1.5 m. The pulp flows on top of the cylinder and falls to the bottom by gravity, the speed of the flow being regulated with a valve. The average product, with pulp from a 60- or 80-mesh screen, is about 45 per cent. of sand and 55 of slime. The slime can be sent to a second set of cones for a more complete separation if desired.

Tube-Mills.—The construction of the tube-mill is so well known as to require no description here. The following notes are taken mainly from operations made at a 40-ton plant at Temascaltepec, State of Mexico, during a run of four months. Additional data are taken from Dos Estrellas and other plants.

The tube-mill at Temascaltepec was 4 m. long, with an inner diameter of 1.2 m. It was operated at from 18 to 27 rev. per min., the best efficiency being obtained with 18. The power at the start was 50 h.p. and 20 while running.

The results, after three months' run, were: Capacity per 24 hr., 56 tons of pulp; production of slime, 14 tons; pebble-consumption per ton crushed, 3.9 kg. The flint liners only began to wear after a run of three or four months; cast-iron liners last much longer. The best result was obtained with heads passing the 60-mesh screens and a 120-mesh discharge-screen.

For further sliming, the outlet of the tube-mill was reduced to 13 cm. by fitting a pipe and valve to control the discharge. It was impracticable to reduce this outlet below 13 cm., since the tube-mill would then get filled entirely and discharge through the inlet.

From the tube-mills the pulp is fed to another series of classifiers, or cones, in which the cyanidation is begun. Either there is a special plant for sand-treatment, or all the ore is slimed to undergo the same treatment.

Cyanidation and Air-Agitation.

Sand-Treatment.—The sand flows directly to the collecting-tank for drainage, which lasts from 36 to 48 hr. This step, however, permits the use of a much weaker solution, and a corresponding economy of cyanide, since the sand-moisture is from 45 to 50 per cent. After drainage, the sand is fed to filter-tanks with a special filter-bottom consisting of a wooden lattice frame-work, covered with a layer of cocoa-matting, over which is stretched ordinary canvas. The depth of these tanks is from 1.2 to 1.5 m.

The sand carries about 16 per cent. of moisture. According to the acidity of the ore, caustic lime is added, amounting to from 2 to 6 kg. per ton.

Each tank, once filled to three-quarters of its height, receives

the weak solution, carrying from 0.25 to 0.30 per cent. of NaCN, generally introduced from the bottom by means of a 5-cm. drop-pipe, terminating in a T. This operation lasts from 6 to 7 hr., then the charge is allowed to soak 6 hr., and filtering is commenced by opening the weak-solution discharge-valve. From time to time, during two or three days, weak solution is added at the top, and at the same time air is introduced by stopping the solution in the pipes, or more effectually with special pipes and an air-compressor. An abundance of air is more necessary with silver-ores containing very little gold than with gold-ores carrying a little silver.

The strong solution is then introduced to the amount of from 60 to 80 tons of 0.75 or 0.80 per cent. of NaCN content, which lasts 48 hr. Later, weak solution is run again as rapidly as possible, and the residue is ready for water-sluicing. The efficiency of air is more complete if a second treatment can be made on a second series of lower tanks.

As the outlay of capital for a sand-treatment plant is very large, it has been necessary to do away with the sand or treat it together with the slime. The new Pachuca system treats both sand and slime together and avoids the necessity of a separation.

The cyanide-consumption is from 1.4 to 2 kg., and the zinc-consumption is 450 g., per ton of sand. The average cost of material per ton treated is but little less than $1 (U. S.), which is a high cost.

Costs.—The total cost of sand-treatment, including freight charges, in similar conditions to those at Temascaltepec, is, in Mexican currency, about as follows:

Cyanide, 1.5 kg.,	$1.50
Zinc,	0.30
Caustic lime,	0.12
Lead acetate and other supplies,	0.28
Labor, salaries, freight,	1.60
Power, lighting, assay-office,	0.70
Management and general expenses (40-ton plant),	0.50
Total,	$5.00 ($2.50 U. S.)

Although the freight-charges are in direct proportion to the distance from the railroad, the mines further south in Mexico have the advantage of lower labor-cost.

The cost of the treatment of the precipitates, including the various taxes of the Mexican government, may be estimated at from $0.80 to $1 (U. S.). In the central part of Mexico crushing costs about $1 per ton, so the total cost of sand-treatment will average about $4.50 (U. S.).

Slime-Treatment.—The treatment of slime during several months at the 40-ton plant at Temascaltepec is as follows:

The plant was equipped with (1) three upper tanks, *A*, 4.8 m. in diameter and 3 m. deep, for the water and the solution; (2) five agitation-tanks, *B*, 6.30 m. in diameter and 4 m. deep; (3) three filter-tanks, *C*, 4.2 m. in diameter and 0.9 m. deep; (4) five lower tanks or sumps, four of which, *D*, were 4.80 m. in diameter and one, *E*, 6.30 m.; each was 3 m. deep.

As shown in Fig. 1, the slimes flowing out from the tube-mill, *t*, are separated in a cone, and then raised 6 m. vertically in *D*, under pressure from a compressed-air injector, *p*, receiving air from the compressor, *m'*.

These slimes, averaging 350 g. of silver, come from ores having an average silver-content of 594 g., 42 per cent. of which was extracted on the concentrating-tables.

From 1,000 tons treated, the recovery by concentration was 243 kg. of silver. The cyanide extraction was 174 kg. of silver, or only 50 per cent. of the remainder, but it is a well-known fact that the losses at the start are always high; it is only by degrees that the difficulties are overcome.

Table I. gives a detailed record of operations made in an agitation-tank. Such records, although far from perfect, are interesting, as it is the first time that cyanidation has been tried with the Temascaltepec ores.

TABLE I.—*Mill Record at Temascaltepec.*

Tank No. 1.	Filling.	Agitation.	Strength of Solution.		Lead Acetate.	Depth of Slimes.	Moisture.	Assay-Value of Solution.
			Before.	After.				
	Hours.	Hours.			Kg.	Meters.	Per Cent.	Grams.
Feb. 15, 1908.	38	24	0.15	0.13	5	1.75	52	365
	......	8	0.14	0.14	......	.. ...	......	263
	... ..	4	0.13	0.13	......	......	......	197
	......	4	0.13	0.12	......	......	......	162
	......	4	0.13	0.12	......	......	......	145
	......	4	......	......	......	......	......	126
	......	4	......	......	......	......	......	.....

The total consumption of cyanide for 1,000 tons was 1,100 kg., and of zinc about 450 kg. The high consumption of cyanide

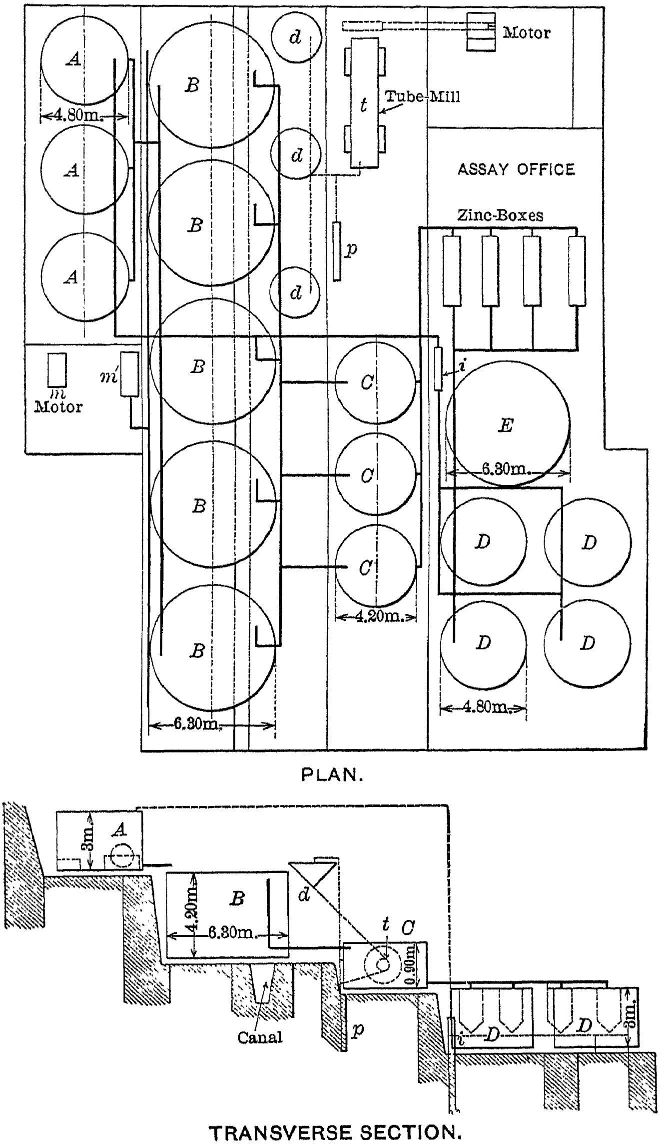


FIG. 1.—SLIME-TREATMENT PLANT AT TEMASCALTEPEC, MEXICO.

resulted from the presence of copper in some of the ores treated; in ordinary ore it would be less than 1 kg. per ton. The lime-consumption was from 3 to 4 kg. per ton.

As regards the above treatment of 1,000 tons, the poor efficiency resulted mainly from the fact that there was still a large proportion of sand mixed with the slime. The proportion of slime was only from 40 to 50 per cent., while at El Oro, for instance, after several years of experience, about 80 per cent. of particles can pass the 200-mesh and 90 per cent. the 150-mesh screen—material which can really be named slimes.

The assay-value of the tailings at Temascaltepec was about 150 g. of silver for slimes averaging 300 g. per ton. The total extraction of 50 per cent. was very weak indeed, due to an insufficient separation of sand and slime, to the presence of copper in some Temascaltepec ores, and to losses of solution.

The compressed air was introduced into the tanks by means of rubber hose and long iron pipes reaching to the bottom and removed from place to place every 10 or 15 min. The agitation-tanks are provided with a wooden bottom to prevent the rapid wearing of the bottom iron sheets; holes have been worn through in many instances. After each period of agitation the pulp settles during from 5 to 8 hours.

As the solution cannot filter through the slime, each agitation-tank is provided with a decantation-pipe and a floating hose with a frame of wood, by means of which the clearest solution of the tank is always supplied to the zinc-boxes. If, by mistake, or neglect, the slimes are allowed to run into the zinc-boxes, it is a great hindrance to the clean-up. This condition is the weak point of the decantation; the filter-tanks are purposely placed below the agitation-tanks to prevent mischief.

The agitation by compressed air has a double effect, oxidizing the mineral particles, which are then ready to dissolve in the cyanide, and breaking up the clay-balls into which the cyanide could not penetrate.

Each agitation-vat is provided with a discharge-opening on the bottom. The pulp, ready for discharge, carrying still about 50 per cent. of moisture, is run through this opening by means of water-pipes and tools. The slimes, falling into a masonry ditch, are run to the waste by the water-current.

There are some improved forms of tanks better fitted for agitation than those used at Temascaltepec. For instance, the Palmarejo tank, which has a conical bottom sloping at 45°, and is provided with a manganese-steel centrifugal pump, running

at a rate of 900 rev. per min., for drawing the solution and pulp continually and mixing with them air admitted through a small valve at the top of the pipes. At Guanajuato and elsewhere the compressed air is run through perforated pipes on the bottom of the tanks; but most of the air runs through the holes near the central air-pipes, while the furthest holes are rapidly shut off by the slimes, which gives very irregular results. Mechanical agitation, however, is preferable to hand-agitation.

An extraction of from 75 to 80 per cent. is reported at Guanajuato by using double treatment and filter-presses. At El Oro, Esperanza, and Dos Estrellas the extraction is from 60 to 70 per cent. of the silver, but the most important value is gold, from 90 to 95 per cent. of which is recovered.

Combined Treatment of Sand and Slime.

Pachuca is again famous because of its new method of treating sand and slime in the tall tanks of the San Francisco mill. The greatest difficulty is to maintain the particles of ore in suspension in the cyanide in order to get a complete extraction. The first idea of using very tall tanks of comparatively small diameter, with an inner cylinder for compressed air, originated in New Zealand. Experiments were made according to that idea at Pachuca, with such a favorable result that the San Francisco mill was erected for a treatment of 100 tons a day. Five additional mills of similar design have since been erected at or around Pachuca.

Grothe Tank.—The Grothe tanks, Fig. 2, are of steel, 4.50 m. in diameter, and 13.50 m. high, including the conical lower part. The tank-capacity is 180 cu. m., the charge varying from 75 to 100 tons. Each tank has a cast-iron discharge-opening at the center of the bottom.

The air-cylinder, kept in position by iron legs, is 0.45 m. in diameter, open at both ends, starting from 0.45 m. above the bottom, and ending 0.45 m. below the upper rim.

An air-pipe, *b*, extends along the axial part of the cylinder. The lower end part is closed and rests upon the bottom, and an air-valve is provided at its inlet into the cylinder to allow escape of air when the pressure is over-balanced by the charge.

A second air-pipe, *a*, in the outer part of the cylinder, is used

to keep the pulp in motion during the filling or the discharge of the tank. A third pipe, *d*, is used for introducing air, water, or cyanide solutions.

Agitation.—The treatment consists in agitating the slimes for about 24 hr. with two or three times the weight of cyanide solution; then a rest of from 6 to 12 hr. The charge, after sufficient agitation, with the addition of slaked lime, is allowed to settle, and the clear liquor is decanted. The slimes are then pumped to the Butters filter-frame tanks for exhausting the last traces of silver-bearing solution.

At the start, the air-pressure is from 3 to 4 kg., but when the circulation is perfect, 2 or 3 kg. is sufficient. The con-

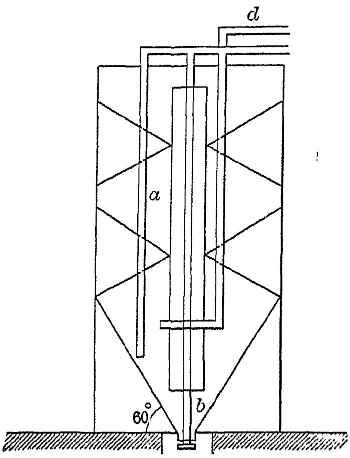


FIG. 2.—OUTLINE SKETCH OF THE GROTHE TANK.

sumption of air is from 3 to 4 cu. m. per min. An air-compressor of 15 h.p. is adequate for a 100-ton plant. The great advantage of the Grothe tank is that it permits the combined treatment of sand and slime; an extraction of 80 to 85 per cent. has been obtained with certain silver-ores. The consumption of cyanide is slightly higher than with the slime-treatment, since a stronger solution is necessary for penetrating the sand.

CYANIDATION OF THE OLD TAILINGS.

There are vast accumulations of tailings in various parts of Mexico, as at Pachuca, Guanajuato, and Zacatecas, which contain silver to the extent of approximately 200 g. per ton, and the treatment at Pachuca apparently presents no special difficulty.

Filter-Presses.

For a total extraction of the silver-bearing solution still contained in the slimes after the usual treatment at Dos Estrellas, El Oro, and other mines, the slimes are passed through filter-presses of large size.

The pressure is about 10 kg. per sq. cm., which makes the slime-cakes quite hard. The general practice is to make two or three washes; the last water then contains only a trace of silver, but a single wash may be sufficient for obtaining the same result. The filter-presses are from 5 to 6 m. long, and from 1.20 to 1.50 m. in diameter, and contain from 60 to 75 filters.

Precipitation.—The following is a brief description of the operations at Temascaltepec.

Each box is fitted with a V-shaped bottom, a discharge-opening, and a valve below it. A 25-mesh screen separates the zinc-shavings from this bottom; zinc-dust is not now used. There are four rows of six-compartment iron boxes, each compartment being 1.20 m. long and 0.90 m. in width and depth.

Since practically all the precipitation takes place in the first three compartments, the zinc is successively taken out and washed on a special wooden box with a metallic screen on top. The bottom screen of each compartment is then taken out, and the whole precipitate is run out through the discharge-valve on the same wooden box as the zinc-shavings. The metallic particles of zinc remain on the upper screen, while the precipitates drop in the box and the water is filtered through the bottom. A special filter-press for precipitates gives a more-rapid result, but in a remote country, and for doing the first cyanide experiments, a wooden box is all that is necessary.

The black precipitate contains from 600 to 700 kg. of silver per ton, according to the proportion of slime that came through the filter. The precipitate is finally dried and either shipped or smelted.

Smelting.

For melting, the precipitate is mixed with 18 per cent. of ground borax-glass, from 6 to 8 per cent. of calcined soda, and a like amount of sand and lime. It takes from 3.5 to 5 hr. to melt and pour one charge. At many mines the slimes are shipped without melting, since milling is economical only if

practiced on a certain scale. Besides, the government tax on the bullion is higher than on the precipitates, which are considered as concentrates.

Costs and Consumption of Material.

Cost Per Ton.—The consumption of sodium cyanide for the treatment of slimes containing from 300 to 350 g. of silver per ton, sulphides, antimonides, and arsenides, does not run much above 1 kg. per ton. The consumption of zinc is 400 g. per ton.

The following cost-data cover a period of three months' operations with a 40-ton plant. The prices are in Mexican currency:

Cyanide, 1 kg.,	$1.00	
Zinc, 400 g. at $0.60,	0.24	
Lime, 4 to 6 kg. at $0.02 max.,	0.12	
Sundries: lead acetate, oil, etc.,	0.28	
Labor and salaries,	1.26	
Assay-office, power, light,	0.70	
Management and general expenses,	0.50	
	$4.10	($2.05 U. S.)
Taxes,		0.90
Crushing,		1.10
Mining,		2.50
Total cost per ton,		$6.55 (U. S.)

The above data correspond to an ore averaging 320 g. of silver per ton (silver quoted $0.65 an ounce). With 200-stamp batteries, it has been possible to reduce the total cost of treatment per ton to slightly below $6 under special conditions.

I have omitted to say anything about cyaniding the concentrates, because the present practice is to ship them direct on account of the high cost of cyanide. The recent progress in cyanidation may lead to the abandonment of the concentration of silver-ores. The tendency is to run the pulp direct from the battery to tube-mills and the all-sliming plant.

Losses in Gold and Silver.

The average extraction by the *patio* process, as indicated by A. H. Bromly, may be taken at from 85 to 90 per cent. of the silver, and *nil* to 25 per cent. of the gold. In the modern

combination method, in which the *patio* is both preceded and followed by concentration, the extraction may reach 95 per cent. of the silver and 84 per cent. of the gold.

In cyanidation, the gold-extraction is from 90 to 95 per cent. The loss in silver depends chiefly upon agitation, and the extraction varies from 60 to 85 per cent. with the all-sliming method and Pachuca tanks.

Cyaniding Slime.

BY MARK R. LAMB, MILWAUKEE, WIS.

(Spokane Meeting, September, 1909.)

THE various methods of treating pulp in air-agitation tanks offer problems for experiment and study which are fascinating as well as practical. The usual method heretofore has been to fill each tank in turn, agitate the mixture the required period and then discharge the treated pulp into a storage-tank, from which it is drawn to the filter as required.

A later method is to run the pulp from the tube-mill, classifier, thickener, or settler into the first of a set of tanks, and thence continuously or in series through the remaining tanks, finally drawing off the pulp to a continuous filter, or to a pulp-storage tank if the filter is of the intermittent type. It is this method which I propose to discuss here in a preliminary way, with the idea of going exhaustively into the examination of the results of a series of experiments in a later paper.

It may be of interest to note that the design of a cyanide-plant has recently been completed in which the estimated cost of the agitation-tanks, storage-tanks, and pumps was 30 per cent. higher for the charge-agitation arrangement than for the series plan.

The first question in studying the series method is, "What is the actual period of treatment undergone by the pulp?" One first supposes that by chance some of the pulp may get through with little or no treatment, while other portions may stay in the tanks a needlessly long period. The former is not probable or even possible, and the latter is, practically, true

to only a limited extent. The greater the number of tanks the greater is the percentage of chance that all the pulp will be treated equally. Without going into the mathematical analysis of the problem (which is of little value, since it is based on problematical premises), its important terms may be clearly grasped by thinking of, or "visualizing," the conditions when, say, 50 tons of dry slime are treated in 24 hr. The thickened pulp representing this tonnage is a sluggish, 2-in. stream flowing 25 in. per second. Taking the flow of this stream for any given period, say 5 min., 16 cu. ft. will flow into the first of a set of agitation-tanks. If the tanks have a working-capacity of 1,600 cu. ft., one such tank will contain 17 tons (dry weight) of slime in a 2-to-1 pulp, and three tanks will be required to agitate 50 tons for 24 hr. Assuming an instantaneous mixture of the stream of incoming pulp with that already in the tank (an impossible condition, which will be discussed later), then 1 per cent. of 16 cu. ft. will be discharged with no treatment. With two tanks the proportion which would be discharged with no treatment is reduced to 0.01 per cent. Of course, other portions would be discharged with varying and increasing periods of treatment.

The rate of flow of the pulp from the time it is thrown out of the top of the central tube until it reaches the bottom of the tank varies with the amount of air used and the shape and the size of the tank. The period is from 0.5 to 2 hr. If the incoming stream enters at the side of the tank opposite the outlet, and giving due consideration to this slow movement of the charge, it is a physical impossibility for any particle of the pulp to go through this series of three tanks in less than 1 hr. 30 min. In the mathematical analysis this minimum period gives the series method a big handicap, and one which increases directly with the number of tanks used. In fact, this low circulation-velocity makes the operation resemble the treatment of pulp in a horizontal tube of the same diameter as the agitation-tanks and of a length equal to the combined lengths, having the agitation at right angles instead of parallel to its length.

If any comparisons have been made on a practical scale to show the difference in extraction by the two methods of using the air-agitation tanks, the results, so far as I know, have not

been published. Such a test would be of great interest, and the managers of several plants are now in a position to make it. In tests on a small scale, with tanks of a capacity of 100 lb. of dry slime in a 2-to-1 pulp, several interesting facts have been noted. With pulp containing coarse sand (and even with fine sand if in a thin pulp), care must be exercised in drawing the pulp from one tank to the next in the series, otherwise either the sand or the slime will accumulate. If the drawing-off pipe is merely flanged to the tank, the outflowing pulp will contain a greater proportion of slime to sand than is contained in the average of the pulp, since during agitation by this method the sand has a tendency to sink through the pulp as soon as discharged through the central air-lift, rather than to work out to the sides of the tank with the flocculent slime. This difference in pulp-flow is greater with a thin mixture and less with a thick one, and can be utilized practically in a very simple manner if it is desired to agitate the sand longer than the slime. A concentration of sand in the tank will be attained by drawing the slime from a quiet point in the circumference of the surface of the pulp. To do this, an inclined discharge-pipe, pivoted at the side of the tank, is arranged so that its inlet end may swing in a horizontal plane and be placed and held at any desired point in its arc between the central pipe and the side of the tank. After the desired initial concentration is attained, this discharge-pipe is then placed so as to draw off a mixture of the same proportions of sand and slime as that being fed to the tank, which will maintain the charge in the tank at the higher ratio of sand to slime. Thus the slime will go through faster and the sand slower than the average pulp-flow.

To illustrate this point, imagine a tank containing 100 tons (dry weight) of pulp consisting of 3 of sand to 1 of slime. Admit a stream of pulp carrying 100 tons in 24 hr., which consists of half sand and half slime, drawing off an exactly similar stream. The slime which was originally in the tank (25 tons) will be replaced twice in 24 hr. by the 50 incoming tons, while the original sand (75 tons) will be replaced only two-thirds time. In other words, the slime will receive a 12-hr. treatment and the sand will receive a 36-hr. treatment. I consider this one of the important advantages of the "series" system. The regulation of the ratio of sand-concentration in

the tanks, within the practical limits to be determined in each case, will be controlled by taking a pulp-sample of a certain size, washing it through a 200-mesh screen, and setting the discharge-pipe as many inches from the rim of the tank as there are ounces of sand remaining on the screen. The proper weight of the sample to give best results can be determined experimentally.

A point of great advantage possessed by the series method in the design of many plants is the saving in difference of level required in the site. The agitation-tanks, if used in this manner, need not require more than from 2 to 4 ft. difference in

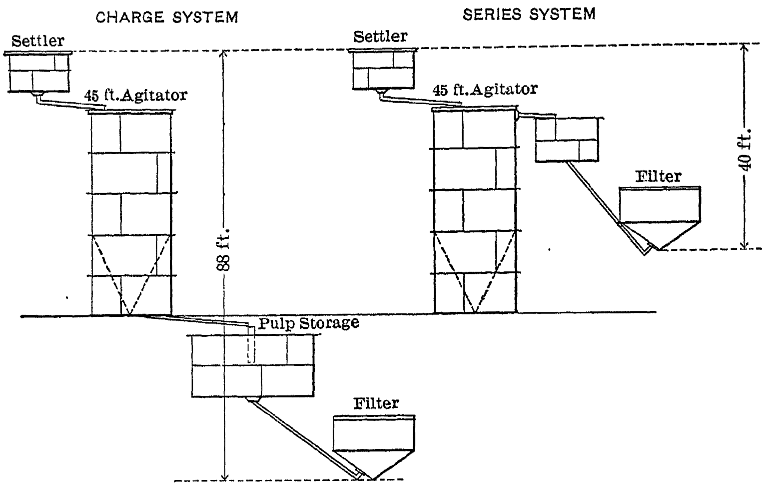


FIG. 1.—CHARGE SYSTEM AND SERIES SYSTEM COMPARED.

level between the inlet and the outlet, depending to some extent upon the number and size of tanks and their distance apart, but also and mainly upon the margin of capacity required to provide for fluctuations in pulp-feed and discharge. This margin depends largely upon the type of filter used. Ordinarily, the pulp will be drawn from near the top of the last tank, while in the "charge" system the outlet is at the bottom of the tank. Fig. 1 shows this difference. In the series system no storage-tank is needed below the last agitator-tank if a continuous filter, such as the Oliver or the Ridgway, is used, only a small tank of a capacity to supply pulp to equal the displacement of a loaded basket of leaves if a Moore filter is used, and

a slightly larger tank (sufficient to fill the filter-box with the unloaded frames in place) with a Butters filter. Compare this arrangement with the charge system, in which the storage-tank, even with continuous filters, must have at least a capacity equal to that of an agitator, and with either the Moore or the Butters filter must have a slightly greater capacity, as is explained above.

The question as to the least number of tanks which can safely be used in this manner, can only be answered by experiments. Even with the charge system, the occasion will rarely arise in which it will be convenient to use less than two tanks for agitation, and in such a case three smaller tanks with equal average treatment-capacity will be so much smaller that their combined cost will be little, if any, more than the cost of the two larger tanks. In a plant having air-agitation operated on the charge system, it is rare that the site will permit all transfers of pulp —from settler through agitators and storage-tank to filter— by gravity. Usually the saving of about 45 ft., as shown in Fig. 1, will practically amount to the saving at least of the cost of a 4- or 6-in. steel-lined centrifugal pump, with its requirement of from 10 to 20 horse-power.

There are numerous plants at which a change to the series method of treatment could be made at slight expense, with an increased capacity and easier work for the plant-foreman. I have in mind a plant with at least twenty 12-ft. air-agitation tanks, each of which is used as a settler in its turn. This arrangement entails continual running by a shift-man and several helpers. The result of installing a Dorr continuous slime-thickener, together with connections between all these agitation-tanks, would be a revelation and a revolution.

The Assay and Valuation of Gold-Bullion.*

BY FREDERIC P. DEWEY,† WASHINGTON, D. C.

(Spokane Meeting, September, 1909.)

THE Bureau of the Mint of the United States Treasury maintains 13 offices for the purchase of gold-bullion, and this paper describes an investigation to establish the reasonable differences in the assay-results at the various institutions which may be commercially allowable in the settlements between them. Beginning with the comparative assay of proof-gold at the Philadelphia mint and the Utrecht mint, which shows 0.00002 as the closest agreement now possible, nine tables of comparative results, taken from the regular work of the service, are given. These tables begin with very fine gold, produced in an electrolytic refinery, showing close agreement in the assay-results, and follow through decreasing gold fineness and increasing amounts and complexity of base metals to very impure and complex bars produced at cyanide-mills, some of which give widely-varying results. Next is given a series of results on samples, prepared and sent out to various laboratories in the service, to test the influence of different metals and various combinations upon the agreement of the assay-results; 11 samples were sent out and each one was assayed from 44 to 71 times, making a total of 623 assays. To these are added 107 assays of identical samples of coin-gold.

On a previous occasion,[1] I have endeavored to show the degree of accuracy that may be expected in the ordinary everyday analysis of various materials, and on another occasion,[2] I

* This paper in full was read at the Seventh International Congress of Applied Chemistry, London, May 27 to June 2, 1909, and was presented at this meeting in advance of its appearance in the *Annual Report of the Director of the Mint for the Year* 1908–1909, through the kind permission of the Director.

† Assayer, Mint Bureau, U. S. Treasury, Washington, D. C.

[1] The Actual Accuracy of Chemical Analysis, *Trans.*, xxvi., 370 (1896); *Journal of the American Chemical Society*, vol. xviii., No. 9, p. 808 (Sept., 1896).

[2] The Accuracy of the Commercial Assay for Silver, *Journal of the American Chemical Society*, vol. xvi., No. 8, p. 505 (Aug., 1894); Accuracy in Silver Assaying, *Journal of the Society of Chemical Industry*, vol. xv., No. 6, p. 434 (June 30, 1896).

have called attention to the accuracy of the commercial assay for silver. The present paper deals chiefly with an effort to establish certain commercial standards of agreement or accuracy in the assaying of gold-bullion for purchase in the various laboratories of the U. S. Mint Service.

Besides the coinage-mints at Philadelphia, New Orleans, Denver, and San Francisco, and the large assay-office on Wall street in New York City, all of which purchase and refine bullion, the Mint Bureau of the U. S. Treasury maintains eight smaller assay-offices, scattered about through the mining-districts of the country, which purchase bullion and ship it to the mints to be refined. These smaller offices were established as an aid to the mining industry of the country by giving the small miners the opportunity to sell their bullion easily and quickly for cash.

Owing to the particular and rigid methods of book-keeping of the Treasury Department, the mints are compelled to treat the bullion sent to them from the assay-offices in exactly the same manner as the bullion deposited directly by individuals, and the prices are carefully determined at which the assay-office bullion should be charged against the mint in the Treasury accounts. Naturally, discrepancies sometimes arise between the mints and the assay-offices. A very large proportion of these are small and are easily adjusted. In fact, most of them adjust themselves automatically, as they are on both sides of the account, and the gains and the losses over a period of time will counterbalance each other. On rare occasions, however, the differences require adjustment by umpire-assays in the laboratory of the Bureau of the Mint at Washington. For several years I have been gathering data upon the subject, and have had a series of assays made in order to establish standard limits of differences which might be considered as allowable on different classes of bullion.

The methods of assaying followed in the various institutions are substantially the same, and have grown up as the result of many years of experience, so that with careful work on pure bullion the results obtained at different institutions ought to agree very closely; but with impure bullion, that is, bullion containing other constituents besides gold and silver, the chances for variations in the results increase. The action of

different impurities varies widely. Only small amounts of some impurities induce excessive variations in the results, while comparatively large amounts of others have but little effect, and, on the other hand, a combination of several impurities in a bullion may be most disastrous to any agreement of the assay-results.

The bullion is handled in the same manner at all the institutions. It is weighed as received, and then melted. Generally, a simple melting with soda or borax, or both, is sufficient, but sometimes it is more or less refined in the pot. In the case of large melts, 1,000 oz. or over, or of very impure bullion, a small sample may be dipped or poured out from the well-stirred pot and granulated in water. The granulations are used for the assay-sample. In general, however, the metal is cast into bars, and these bars are chipped, top and bottom, to obtain the assay-samples. The bars are again weighed and the assays made, when the value of the deposit is calculated from these data. If, however, the various assays made on a deposit do not agree well enough to satisfy the assayer, the bar is remelted, with or without refining in the pot.

The determination of gold in ores by the fire-assay, when properly executed, is justly regarded as one of the most accurate of analytical methods. With ordinary care and an excellent bead-balance, 1 part of gold in more than 20,000,000 parts of ore can be readily and accurately determined. The determination of 1 part of gold in 5,000,000 parts of ore is very easily done. Until recently, however, it was rare for commercial ore-assaying to attain to the accuracy of 1 part in 5,000,000.

The ability to determine gold in ores with such great accuracy is due to the fact that very large amounts of ore, up to 0.25 kg., are taken for the assay, and on a high-grade button-balance the resulting bead can be weighed to 1/200 mg. In assaying bullion, however, such extreme accuracy is out of the question, for the simple reason that there is a limit to the amount of bullion that can be taken for the assay. To obtain the most accurate results the assay-sample must be weighed on the same high-grade balance on which is weighed the resulting cornet, and the sample also must be weighed with the same degree of care and accuracy as the cornet. Now, the load that

a very sensitive bead-balance will safely carry is generally limited to 1 g., and the amount of metal generally taken for a gold-bullion assay is 0.5 g., or one-half of the maximum load of the balance.

Another point in bullion-assaying which militates against extreme accuracy in the results lies in the fact that the cornet which is weighed is itself gold, and, in high-grade bullions, it is a very large part of the sample taken for the assay, so that even slight errors in the handling of the cornet, resulting in slight losses or gains in its weight, count heavily against the highest accuracy of the results.

About two years ago samples of proof-gold were exchanged between the Philadelphia mint and the Utrecht mint, and these samples were assayed in comparison with the utmost care at both institutions, with the result that the Utrecht proof was pronounced slightly the better by both mints. The difference in the results of the assays at the two places was only 0.00002. This is by far the most careful and exhaustive comparison of gold-bullion assays known to me, and undoubtedly represents the limit of accuracy at present attained in such work.

Table I. shows a series of results obtained by three assayers working in the same laboratory upon fine gold from an electrolytic refinery. Each assayer worked upon the same sample in each set of assays as averaged, the samples being cut from both the top and the bottom of the bars. While there is a possibility that there may be some difference in composition between the top and bottom of the bars, yet in such high-grade material as this any such difference must be slight, and eight tests upon the subject showed a maximum difference between the top and bottom of only 0.0001, which is considerably less than many of the differences between individual assays. On the whole, then, the figures may be taken as fairly representing the ordinary run of commercial work upon such high-grade bullion. It will be noted that in several cases the figures exceed 1,000, which is due, in part at least, to the high grade of the material. It may also be due in part to the presence in this electrolytic gold of unusual impurities in very small amounts. These data emphasize the necessity of averaging a large number of assays to get a satisfactory determination of the fineness in such very high-grade material.

TABLE I.—*Fine-Gold Assays.*

1.	2.	3.	Average.	1.	2.	3.	Average.
999.8	999.6	999.6		999.7	999.7	999.3	
999.5		1000.0	999.7	999.5		999.8	999.6
999.7	999.5	999.4		999.9	999.8	999.5	
999.8		1000.1	999.7	999.8		1000.0	999.8
999.7	999.5	999.6		999.6	999.7	999.6	
999.8		1000 3	999.8	999.9		1000.3	999.8
999.4	999.4	999.6		999.7	999.5	999.7	
999.7		1000.3	999.7	1000.0		1000.4	999.8
			999.7				999.8
	999.5	999.6			999.8	999.9	
1000.1		999 9	999.8	1000.0		1000.2	999.9
999.8	999.9	999.6		999.7	999.8	999.8	
999.5		1000.1	999.8	999.5		1000.3	999.8
999.8	999.8	999.7		999.9	999.9	999.6	
999.5		1000.2	999.8	999.7		1000.1	999.8
999.9	999.7	999.6		999.8	999.8	999.8	
1000.0		1000.3	999.9	999.9		1000.5	999.9
				999.8	999.9	999.7	
				1000.1		1000.4	999.9
				999.8	999.8	1000.0	
				1000.1			999.9
			999.8				999.9

Table II. shows results obtained by various assayers in a single laboratory in assaying granulation-samples from a wide variety of bullion.

The figures given in Table III. are all taken from a single shipment and show the accuracy that can be obtained upon material of fairly uniform composition, being mostly gold and silver, with but little base metal present. This table shows, first, the results obtained at the assay-office where the bullion was originally purchased; and, second, the results obtained upon the same material when shipped to a mint. In some of these samples there is undoubtedly a difference between the tops and bottoms of the bars, but the figures show the agreement that may be expected between two institutions in arriving at the value of such deposits.

Table IV. gives the assays of 14 bars which were referred to the Bureau laboratory for adjustment, although the average differences between the mint and the assay-office were only slight.

The handling of bullion produced at mills using the cyanide-process of gold-extraction has given a great deal of trouble.

Table II.—*Miscellaneous Gold-Assays.*

Gold Fineness.				Silver Fineness.
0.4	0.4	0 3	0.1	997.5
0.5	......	0.4	0.4	862.5
2.8	2.8	2.8	2.9	955.0
6.1	6.2	6.3	6.3	970.0
11.0	10.9	11.1	10.9	888.0
12.0	12.1	12.3	12.3	805.0
17.0	16.5	17.1	17.0	967.5
19.4	19.4	19.7	19.6	835.0
29.4	29.4	29.2	29.2	709.0
36.1	36.0	35.9	36.0	689.0
43.2	43.2	43.1	......	47.0
47.0	46.4	45.4	46.3	304.0
52.4	51.8	52.0	51.4	79.0
62.5	62.7	62.1	62.5	766.0
68.0	68.2	68.3	68.0	362.0
79.0	79.4	79.0	79.1	731.0
108.5	109.3	108.7	108.9	495.0
148.3	148.4	148.3	148.4	372.0
179.0	179.1	179.0	....	695.0
194.1	194.3	194.7	195.0	771.5
208.3	208.3	208.6	....	416.0
308.4	308.8	308.5	308.6	149.0
439.9	440.0	439.8	440.0	190.0
510.1	510.0	509.6	509.6	236.0
515.0	515.1	514.9	515.2	171.0
537.7	537.8	536.8	537.1	227.0
571.6	570.4	571.4	571.0	185.0
605.3	606.9	606.7	606.8	129.0
642.6	643.0	643.8	642.7	257.0
711.2	710.2	710.7	711.7	3.0
716.0	716.1	716.0	715.9	222.0
758.9	759.0	759.0	759.1	216.0
870.6	870.5	870 2	871.4	27.0
978.0	978.4	978.0	....	17.0

Table III.—*Assays of a Single Shipment.*

	Assay-Office.	Mint.	Assay-Office.	Mint.	Assay-Office.	Mint.	Assay-Office.	Mint.	Assay-Office.	Mint.
Gold fineness.	843.4	843.0	860.9	860.3	862.5	862 6	863.6	863.4	864 8	864 6
	843.4	843 0	860.9	860.6	862.3	862.5	863.6	863.5	864.8	864.6
	843 4	843.2	860.8	860.6	862.5	862.5	863 6	863.4	865 0	864.5
	843.2	843.1	860.9	860.7	862.6	862.6	863.3	863.6	864.7	864.4
	843.2	843.0	860.9	860.7	862.5	862.5	863.6	863 6	864.8	864.9
	843.1	843.3		860.7	862.5	862.6	863,6	863.6	864 7	864.7
Silver fineness...	151.5		135		134		132		131	

	Assay-Office.	Mint.	Assay-Office.	Mint.	Assay-Office.	Mint.	Assay-Office.	Mint.	Assay-Office.	Mint.
Gold fineness.	870.5	870.8	873.5	873.2	874.9	874.9	878.2	878.1	880.1	879.9
	870.6	870.9	873.6	873.9	874.9	874 9	878.3	878.2	880.1	880.0
	870.6	870.9	873.6	873.6	875.1	874.7	878.3	878.3	880.2	880.2
	870.5	870.6	873.4	873.7	874.9	874.7	878.3	878.2	880.2	880.1
	870 6	870.8	873.5	873.5	874.7	874.7	877.8	878.2	880.0	880.2
	870.6	870.8	873.7	873.4	874.9	874.7	878.2	878.1	880.0	880.1
Silver fineness......	125		122		120.5		117		116.5	

TABLE IV.—*Comparison Between Assay-Office, Mint, and Bureau.*

	Gold Fineness.													
Assay-office.	736.6	807.5	850.2	853.1	866.8	868.3	876.4	878.2	879.1	884.0	886.0	892.8	897.4	899.5
	736.4	807.6	850.3	853.4	867.0	868.0	875.7	878.2	879.1	884.1	885.5	892.5	897.3	899.2
	736.5	....	....	....	866.5	868.4	....	....	....	884.0	....	....	....	....
	736.5	....	....	....	866.6	868.6	....	....	....	883.9	....	....	....	....
Mint.........	735.4	807.2	849.1	852.6	865.7	867.5	875.1	877.6	878.3	883.7	885.0	891.7	896.4	898.8
	735.4	806.7	849.4	852.7	865.2	866.9	875.4	877.6	878.6	883.0	885.4	891.9	896.2	897.8
	735.1	807.3	849.9	852.5	865.9	867.5	....	877.7	878.5	883.9	885.2	891.7	896.6	898.9
	735.9	807.1	850.2	852.5	866.3	867.5	....	877.8	878.8	883.6	885.3	892.0	896.8	898.0
Bureau.......	735.9	807.4	850.1	852.7	866.8	867.9	875.9	878.2	879.2	883.7	885.7	892.6	897.2	898.7
	736.0	807.5	850.1	853.0	866.6	867.5	875.9	878.1	879.2	883.7	885.5	892.8	897.1	898.7
	736.1	807.5	850.2	853.1	866.7	867.8	875.8	878.3	879.4	883.8	885.9	892.1	897.1	898.6
	736.1	807.5	850.0	852.9	866.8	867.7	875.9	878.4	879.1	883.6	885.9	892.1	897.3	898.8
Highest..........	736.6	807.6	850.3	853.4	867.0	868.6	876.4	878.4	879.4	884.1	886.0	892.8	897.4	899.5
Lowest...........	735.1	806.7	849.1	852.5	865.2	866.9	875.1	877.6	878.3	883.0	885.0	891.7	896.2	897.8
Difference........	1.5	0.9	1.2	0.9	1.8	1.7	1.3	0.8	1.1	1.1	1.0	1.1	1.2	1.7
Silver fineness..	179.0	181.0	133.0	142.0	127.0	112.0	120.0	117.0	115.0	113.0	109.0	88.0	99.0	95.0

Even when properly prepared such bars are likely to be troublesome, but when, as not infrequently happens, the precipitates are not properly purified before being cast into bars they may give no end of trouble.

A very mild case of variation in cyanide-bars is shown in Table V. As received, these bars were chipped and the chips assayed. Since the figures thus obtained were considerably higher than the shipper's figures, the bars were then carefully bored and the borings assayed. Finally, the bars were remelted, with small losses in each case, and granulations taken. The granulations were then assayed.

TABLE V.—*Assay of Cyanide-Bars.*

	Gold Fineness.				
Chips.......... ...	394.1	381.6	380.7	381.6	440.8
	392.7	383.2	381.7	381.8	440.9
	392.0	381.3	381.5	381.5	440.3
	392.0	383.4	383.4	382.8	440.4
Borings...........	394.0	381.6	380.7	381.6	440.8
	392.1	381.3	381.5	381.5	440.3
Granulations.....	393.5	382.6	382.0	381.8	440.7
	393.4	382.3	382.7	382.5	440.4
	393.6	382.8	382.4	381.2	440.1
	393.8	382.2	383.2	381.4	440.1
Silver fineness........	370	370	370	370	357

Table VI. exhibits the results obtained by sampling three cyanide-bars, high in gold and very low in silver, in three different ways. The assays show a wide variation on the chip-samples. While the drill-sample assays are fairly concordant for this class of material, the dip-sample assays agree much better and are to be preferred.

An assay-office had received a cyanide-bar which showed 546, 545.5, 546.2, 546 fine in gold. This was considered satisfactory, and it was shipped to a mint, but the chip-samples there yielded most varying results, as follows: 544.6, 535.2, 543, 535, 542.4, 555.6. The bar was then remelted, and granulations showed 550.2 and 551.2. Another cyanide-bar received at the same assay-office from the same mill showed 592, 593.9, 592.9, 593.3 fine in gold, and was accepted. It was shipped to the same mint, where chips showed 603.6 and 590, while borings showed 588 and 588.6. The bar, which weighed 559.65

TABLE VI.—*Assay of Cyanide-Bars.*

Sampled in three ways.

	Gold Fineness.		
Chips	833.1 828.1 830.7 842.1	864.2 863.3 866.2 869.6	839.8 841.9 845.2 839.1
Drills	834.5 833.7 832.7 835.5 835.4 834.6	864.2 864.2 865.8 867.4 867.0 866.6	845.5 845.3 845.0 845.4 844.4 845.5
Dips	834.1 834.8 834.2 835.3 834.8 834.1 834.5 834.7 834.6 834.5	865.9 866.6 867.1 865.1 865.2 866.9	845.7 845.1 844.9 845.5 845.0 843.4
Silver fineness	5.5	8.	8.
Weight	1169.06 oz.	1228.40 oz.	1171.16 oz.

oz. Troy, was remelted, with a loss of 1.78 oz., and granulations from the melt showed 601.8 and 601.8 fine in gold.

Having had a great deal of trouble with some bars from this mill, while others gave but little trouble, the assay-office gave one of the bad bars a very thorough treatment by melting and refining in the pot. As received, the bar weighed 643.30 oz. Troy, and was probably about 847 fine in gold. It was melted seven times, when it weighed 502.01 oz., showing a loss of 141.29 oz. in weight. The final bar was 933.4 fine in gold and 21 fine in silver. The gold-loss from this excessive course of meltings was only approximately 3.75 oz., most of which could undoubtedly be recovered from the slags.

The details of the meltings are shown in Table VII. It should be noted that the fourth melt shows practically no refining, and the weight was only slightly reduced, so that no practical change is shown in the assays.

From an extensive series of tests made at the San Francisco mint it was found that, as a rule, in the cyanide-bars from sev-

TABLE VII.—*Cyanide-Bar, Melted Seven Times.*

Original weight, 643.3 oz. Troy.

Melt	Gold Fineness.	
First melt, 557.22 oz.....	847.0	847.0
	847.2	846.6
	848.0	846.3
	847.8	844.6
	847.6	847.6
Second melt, 544.46 oz..	868.1	868.3
	867.1	868.8
	865.8	866.7
	866.1	866.6
	866.8	867.5
	869.2	869.1
	867.9	867.4
Third melt, 536.44 oz...	877.3	877.3
	877.3	879.4
	875.7	876.6
	873.8	875.8
	877.4	878.7
	876.7	877.3
	876.6	876.8
	875.4	877.5
	878.9	874.6
	879.2	874.1
	876.0	865.7
	875.4	875.2
	877.0	878.9
	877.8	879.4
	863.4	877.7
	875.7	876.9
Fourth melt, 535.55 oz..	878.3	878.7
	879.0	871.4
	877.5	861.6
	877.1	878.0
	875.8	879.5
	870.2	879.0
	876.1	877.5
	876.9	878.0
	871.7	875.7
	879.0	879.0
Fifth melt, 511.88 oz....	916.8	917.3
	916.9	917.6
	917.6	916.8
	917.1	916.9
Sixth melt, 504.82 oz...	928.6	928.6
	929.2	928.8
	929.4	928.6
	930.0	928.8
Seventh melt, 502.01 oz.	933.5	933.3
	933.2	933.4
	933.3	933.7
	933.7	933.4

eral California plants, the chip-samples taken from the outside of the bars would be about 2.5 fine less in gold than the borings when taken away from the edges of the bar, and that the borings gave satisfactory samples of the bars.

Thirteen miscellaneous deposits were united in a mass melt and cast into 17 bars weighing 2841.77 oz. Each bar was chipped twice, and each chip was assayed in duplicate for gold. Table VIII. shows the number of times the stated fineness was obtained:

TABLE VIII.—*Mass-Melt Assays.*

Fineness.	Number.	Fineness.	Number.	Fineness.	Number.	Fineness.	Number.
405.0	...	406.0	3	407.0	8	408.0	5
405.2	...	406.2	5	407.2	6	408.2	4
405.4	1	406.4	3	407.4	8	408.4	3
405.6	...	406.6	7	407.6	2	408.6	1
405.8	4	406.8	3	407.8	5	408.8	

The average of the 68 assays showed the mass to be 407.16 fine in gold.

When made from the highest grade of metals our coin-gold, 900 gold and 100 copper, does not segregate. The gold used may contain a very small amount of silver, but should be as free as possible from all other impurities, and the copper should be of the highest purity possible. Occasionally, in practice, however, there will be a slight segregation due to some impurities present in minute amounts. On one occasion an inside strip cut from a double eagle was assayed six times and yielded the following gradually decreasing figures: 900.2, 900.1, 899.9, 899.85, 899.5, 899.45. On another occasion a double eagle was cut as indicated in Fig. 1 and the following results were obtained:

	A.	B.	C.	D.
	Gold Fineness.	Gold Fineness.	Gold Fineness.	Gold Fineness.
Bureau ..	899.12 899.45 899.20 899.45	900.47 900.38	899.89 899.85	899.70 899.58
Mint......	899.4 899.4 899.5 899.5	900.2 900.5 900.2		

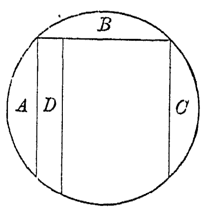


FIG. 1.—GOLD COIN SAMPLED FOR ASSAY.

One of our most annoying and yet very interesting and instructive cases was a lot of foreign-coin gold, the product of a mint which is very careful in the manufacture of its coins; 12 deposits of this material were received at the Philadelphia mint from the New York assay-office. It was supposed to be 916⅔ fine in gold, the balance being copper, and very uniform in composition, but the New York assays showed considerable variation. At Philadelphia one man assayed each deposit in duplicate, and he was checked by another man with a single assay, as shown in Table IX.

TABLE IX.—*Foreign-Coin Gold-Assays.*

	Gold Fineness.											
First assayer.......	917.6	916 4	916.4	917.6	917.5	917 1	915 6	917.3	917.6	914.7	917.0	917.
	918.3	917.9	917.9	917 8	917.9	917.5	917.3	917.3	917.5	916.8	916.9	917.
Second assayer...	917.2	917.3	917.7	917.3	916.5	916.3	917.2	916.3	916.0	915 8	915.4	916.0

This bullion was diluted with copper to bring it down to the United States standard. While our standard is 900 fine, and the law allows a variation of one one-thousandth up or down, so that legally the coins may run from 899 to 901 fine in gold, yet the working-limits adopted at the mints are much narrower than the legal limits, and generally no gold ingots are passed by the assayer below 899.7 fine nor above 900.2 fine.

In making ingots from this metal an unusual number of melts had to be rejected and remelted for want of uniformity. It was expected that the coins made from this bullion would run low, but none of those regularly tested did. In fact, two from one delivery were most unusually high, viz.: 900.6 and 900.7. Thereupon 12 coins were selected from the same delivery and 46 assays were made upon them, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
3 assays showed . .	899.7	4 assays showed . .	900.0	5 assays showed . .	900.4
3 assays showed . .	899.8	7 assays showed . .	900.1	4 assays showed . .	900.5
5 assays showed . .	899.9	3 assays showed . .	900.2	1 assay showed . .	900.7
		11 assays showed . .	900.3	—	
				46	

The trouble with this metal undoubtedly arose from the presence of a small amount of some impurity causing a segregation of the gold, but enough work to decide what this was could not be given to the matter. In a similar case, with a different high-grade foreign-coin gold at the San Francisco mint, the trouble was traced to the presence of a minute amount of antimony.

In order to get a much wider range of comparison, and to test the influence of the different metals and of various combinations upon the gold-assay, a series of samples was prepared in the Bureau laboratory and sent out to various laboratories in the service for assay. In preparing the samples the metal was thoroughly mixed by stirring when molten and remelted as often as appeared necessary. They were finally cast into small bars, and when sufficiently ductile were rolled out thin.

The strips were cut into small squares, and these were mixed up and the samples for each institution taken out of the mixed pile of pieces. In the case of the brittle bars, they were hammered out and rolled until they crumbled to pieces. The larger pieces were then cut up, and the whole mixed before the samples were taken out.

All through the preparation of the samples very great care was exercised, so that in each set every sample sent for assay should be identical, and thus eliminate from the assay-results all chances of differences being due to differences in the samples operated upon, and to confine the differences shown to the actual assay-work. In one very base sample, which will be further noted, it was not possible to adhere to this rule because the metal was too hard.

In making such small melts it is practically impossible to adhere to any predetermined composition with any degree of closeness.

The first sample sent out was gold about 105 fine in silver and about 10 fine in copper; 71 assays of this sample were made in nine laboratories, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
3 assays showed . .	884.1	6 assays showed . .	884.4	7 assays showed . .	884.8
3 assays showed . .	884.2	11 assays showed . .	884.5	3 assays showed . .	884.9
5 assays showed . .	884.3	14 assays showed . .	884.6	—	
		19 assays showed . .	884.7	71	

The averages obtained in the different laboratories were:

Gold Fineness.	Gold Fineness.	Gold Fineness.
884.271	884.517	884.663
884.433	884.517	884.738
884.438	884.631	884.788

A sample approximately 500 fine in silver, 110 fine in copper, and 50 fine in lead was assayed 64 times in nine laboratories, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
2 assays showed . .	340.9	3 assays showed . .	341 4	3 assays showed . .	341.9
7 assays showed . .	341.0	7 assays showed . .	341.5	4 assays showed . .	342.0
11 assays showed . .	341.1	9 assays showed . .	341.6	1 assay showed . .	342.1
7 assays showed . .	341.2	3 assays showed . .	341.7	—	
2 assays showed . .	341.3	5 assays showed . .	341.8	64	

The averages obtained in the different laboratories were:

Gold Fineness.	Gold Fineness.	Gold Fineness.
341.016	341.163	341.600
341.038	341.467	341.863
341.150	341.520	341.913

Two samples were both about 25 fine in mixed base metals, while one was approximately 360 fine in silver, and the other was about 450 fine in silver. The first sample was assayed 61 times in nine laboratories, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
1 assay showed . .	617.6	8 assays showed . .	618.1	4 assays showed . .	618.5
4 assays showed . .	617.7	6 assays showed . .	618.2	9 assays showed . .	618.6
3 assays showed . .	617.8	9 assays showed . .	618.3	2 assays showed . .	618.7
9 assays showed	618.0	6 assays showed . .	618.4	—	
				61	

The averages obtained in the different laboratories were:

Gold Fineness.	Gold Fineness.	Gold Fineness.
617.725	618.233	618.388
618.025	618.283	618.467
618.138	618.320	618.480

The second sample was assayed 60 times in nine laboratories, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
4 assays showed . .	528.6	4 assays showed . .	529.0	7 assays showed . .	529.4
7 assays showed . .	528.7	7 assays showed . .	529.1	2 assays showed . .	529.5
3 assays showed . .	528.8	12 assays showed . .	529.2	1 assay showed . .	529.6
3 assays showed . .	528.9	10 assays showed . .	529.3	—	
				60	

The averages obtained in the different laboratories were:

Gold Fineness.	Gold Fineness.	Gold Fineness.
528.671	529.175	529.267
528.800	529.238	529.283
528.963	529.250	529.300

Having on hand some ferruginous bullion, I attempted to prepare a sample for this work, but experienced considerable difficulty in getting a satisfactory metal, owing to the separation of magnetic globules on solidification. By melting several times with niter I finally obtained a sample that did not

show visible segregation, and it must have been close to saturation with iron. It was about 763 fine in gold and 185 fine in silver, so that the entire base metals, including the iron, were only about 52 fine.

This sample was assayed 47 times in nine laboratories, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
2 assays showed . .	762.9	6 assays showed . .	763.4	4 assays showed . .	763.8
5 assays showed . .	763.0	4 assays showed . .	763.5	5 assays showed . .	763.9
3 assays showed . .	763.2	2 assays showed . .	763.6	2 assays showed . .	764.0
5 assays showed . .	763.3	9 assays showed . .	763.7	—	
				47	

The average results obtained in the different laboratories were:

Gold Fineness.	Gold Fineness.	Gold Fineness.
762.975	763.417	763.700
763.175	763.467	763.683
763.300	763.500	763.833

It having been supposed that much of the difficulty with cyanide gold bars was due to the zinc left in the slimes and going into the bars, a sample was prepared which was nearly 590 fine in gold, about 245 fine in silver, slightly over 130 fine in zinc, and containing a little copper and very little lead.

This sample was assayed 50 times in eight laboratories, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
1 assay showed . .	588.9	7 assays showed . .	589.3	2 assays showed . .	589.7
3 assays showed . .	589.0	7 assays showed . .	589.4	5 assays showed . .	589.8
4 assays showed . .	589.1	9 assays showed . .	589.5	3 assays showed . .	589.9
3 assays showed . .	589.2	6 assays showed . .	589.6	—	
				50	

The average results obtained in the different laboratories were:

Gold Fineness.	Gold Fineness.	Gold Fineness.
589.040	589.417	589.567
589.400	589.475	589.800
589.400	589.483	

A simple inspection of these results shows very clearly that zinc alone does not materially militate against agreement in the

assay-work itself, and if it is the cause of the trouble with cyanide-bars it must be owing to its causing segregation, and thus preventing the proper sampling of the bars by chipping or boring. Other elements may also be active in producing segregation in such bars, either by themselves or through combinations with the zinc or other metals present. A low-grade and very base bar along this line was prepared to run about 100 fine in zinc, 200 fine in copper, and 50 fine in lead.. It was about 268 fine in gold and 370 fine in silver. This bar was very hard, and it was impossible to prepare identical samples for the various laboratories. It was simply cut into pieces and a piece sent to each institution.

This sample was assayed 44 times in eight laboratories, and while the difference between the highest and the lowest result is only 1.7 fine, yet the results are scattered all along through the range, and there is only a slight concentration of the results about one point. This is, of course, due in part to the fact that the samples assayed were not identical.

The results obtained were:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
1 assay showed . .	268.0	3 assays showed . .	268.5	3 assays showed . .	269.1
3 assays showed .	268.1	3 assays showed . .	268.6	1 assay showed . .	269.3
6 assays showed . .	268.2	3 assays showed . .	268.8	3 assays showed . .	269.6
6 assays showed . .	268.3	1 assay showed . .	268.9	3 assays showed . .	269.7
4 assays showed . .	268.4	4 assays showed . .	269.0	—	
				44	

It has long been known in a practical way that the presence of arsenic in a gold-bullion prevents any agreement in the assays. Fortunately, however, the presence of arsenic very plainly reveals itself in the melting of the bullion, and when found the melter proceeds to refine the bullion in the pot, and ultimately removes it very completely before the bullion can be accepted.

Three test-samples containing arsenic were prepared, and they yielded most astonishing and interesting results. The first sample was approximately 785 fine in gold, 107.5 fine in silver, 100 fine in copper, and 7.5 fine in arsenic. This is only a small proportion of arsenic, and yet it completely prevented any agreement whatever in the assay-results. This sample was assayed 65 times in ten laboratories. The lowest result obtained

was 779.7 fine in gold, and the highest 792.4, with an extreme difference of 12.7 in the fineness. Moreover, there is the utmost divergence in the results as well as no agreement whatever; 30 results were obtained only a single time each, 11 only twice each, 3 only three times each, and only a single result was obtained four times. In only three instances did one laboratory obtain the same result twice.

A sample approximately 675 fine in gold, 200 fine in silver, 100 fine in zinc, 24 fine in lead and copper, and only 1 fine in arsenic, yielded a slightly better set of results, but still very widely divergent. This sample was assayed 62 times in ten laboratories. The lowest result obtained was 671.4 fine in gold, and the highest 681.4, showing an extreme difference of 10 in the fineness; 31 results were obtained a single time only, 10 only twice each, 2 only three times each, and only a single result was obtained five times. In three instances one laboratory obtained the same result twice, and in one case a laboratory obtained the same result three times.

It would appear, however, that the influence of arsenic upon the assaying of high-grade bullion containing only trifling amounts of base metals is far less injurious. While the results on a sample running approximately 865 fine in gold, 130 fine in silver, 1 fine in arsenic, and only 4 fine in other base metals cannot be considered satisfactory, yet they are very much better than those yielded by the other two arsenical bullions. This sample was assayed 53 times in nine laboratories, with the following results:

	Gold Fineness.		Gold Fineness.		Gold Fineness.
1 assay showed . .	864.1	4 assays showed . .	865.2	3 assays showed . .	865.9
1 assay showed . .	864.3	8 assays showed . .	865.3	5 assays showed . .	866.0
2 assays showed . .	864.4	2 assays showed . .	865.4	2 assays showed . .	866.1
2 assays showed . .	864.7	4 assays showed . .	865.5	3 assays showed . .	866.2
2 assays showed . .	864.8	1 assay showed . .	865.6	1 assay showed . .	866.6
2 assays showed . .	865.0	3 assays showed . .	865.7	—	
3 assays showed . .	865.1	4 assays showed . .	865.8	53	

The averages obtained in the different laboratories were:

Gold Fineness.	Gold Fineness.	Gold Fineness.
864.933	865.233	865.500
865.183	865.286	865.517
865.200	865.300	865.717

As in so many other directions, antimony behaves similarly to arsenic in assaying gold-bullion, but its influence is not so pronounced. A sample of bullion approximately 723 fine in gold, 245 fine in silver, 1 fine in antimony, and 31 fine in mixed base metals, copper, lead, zinc, was assayed 46 times in nine laboratories. The lowest assay obtained was 721.3, and the highest 725.1, showing a range of 3.8 in the fineness. However, 24 of the results, or just over a half, ranged from 722.8 to 723.9 fine, and outside of this range only two results were obtained more than a single time.

Finally, some of our gold coin was melted up and assayed 107 times on identical samples in five laboratories, with the following results:

	Gold Fineness.		Gold Fineness.
6 assays showed . . .	899.6	32 assays showed . . .	900.0
10 assays showed . . .	899.7	5 assays showed . . .	900.1
26 assays showed . . .	899.8	—	
28 assays showed . . .	899.9	107	

The actual average of this sample is 899.879 fine in gold.

With these results as a basis, the investigation of the subject is being continued with the hope of ascertaining the causes of the variations shown and improving the agreement in the results attained. It is, for instance, well known that gold cornets are not pure gold. They always carry some silver, and I have never failed to find copper in them when tested for with great care. On several occasions I have found lead present on testing the silver nitrate solution from parting a large number of cornets at one time in a platinum basket. The amounts of these base metals present in the cornets are, of course, quite small, and their influence is corrected by the proof-assay, in the same way that it corrects for the silver left in the cornets. I am, however, carrying on a series of quantitative determinations of base metals present in gold cornets, the results of which I hope to publish at some future date.

*

The Nicola Valley Coal-Field, British Columbia.

BY MILNOR ROBERTS,* SEATTLE, WASH.

(Spokane Meeting, September, 1909.)

THE Nicola Valley coal-field is small, but it seems likely to become important because of its commanding position in a rich region that is developing rapidly. Bituminous coking-coal in workable quantity has not been found at any other point between the Crow's Nest Pass region, about 400 miles to the east, and Vancouver island, 250 miles westward. The Canadian Pacific railway reached the Nicola valley two years ago; two other lines, the Canadian Northern and the Great Northern, are building towards the valley. The vast region between the Rocky mountains and the Pacific coast, extending northward for several hundred miles from the international boundary, which has been the scene of extensive mining- and smelting-operations for many years, is filling up with settlers, who are attracted by the wealth of minerals and timber, the rich soil of the valleys, and the fine climate.

The Nicola river, flowing westward through the valley of that name in the Yale district, discharges into the Thompson river, which is the eastern branch of the Fraser. The transcontinental line of the Canadian Pacific railway crosses the mouth of the Nicola river at Spence's bridge, 178 miles from Vancouver. From Spence's bridge the Nicola branch line ascends the valley for 40 miles to Merritt, the coal-center, and continues 7 miles farther to the town of Nicola. Thus the haul from the mines at an elevation of 2,000 ft. to salt water is 218 miles down river-grades. The Nicola branch will be continued about 150 miles farther eastward to Penticton, where it will connect with an extension of the line through the rich Kootenay and Boundary districts. The Canadian government has granted a subsidy of $6,000 per mile, to which the provincial government is planning to add $5,000 per mile, to assist in the construction of the Nicola extension. Upon the

* Dean, School of Mines, University of Washington.

completion of this connecting link the Canadian Pacific railway will have a short southern route from the coast to its main line at Dunmore, Saskatchewan. A map of the coal-field and the railroads already constructed or proposed is given in Fig. 1.

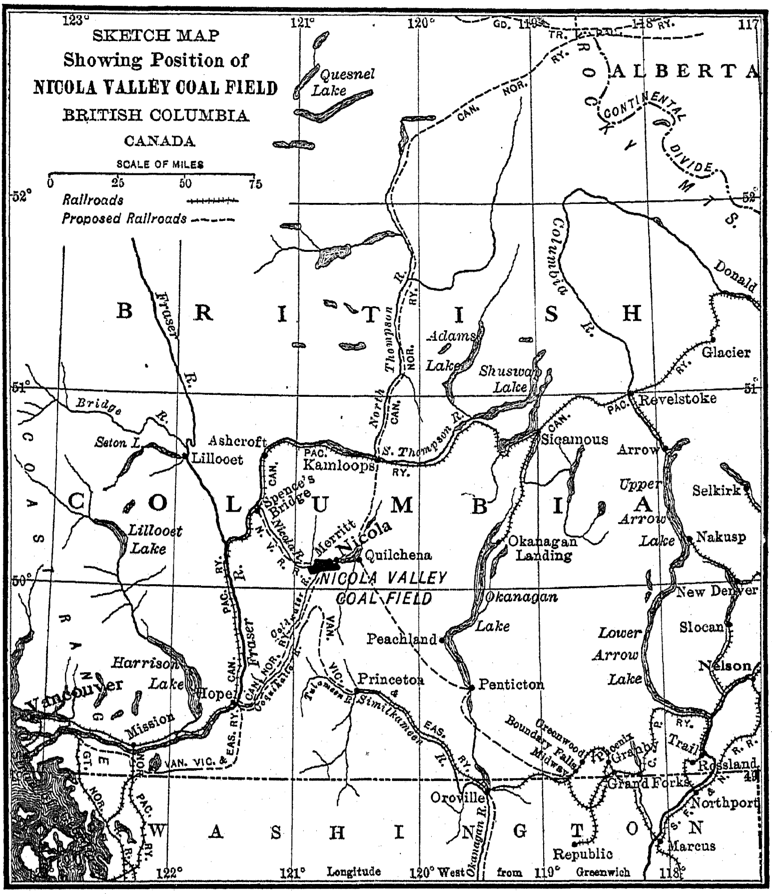


FIG. 1.—SKETCH-MAP OF A PORTION OF BRITISH COLUMBIA, SHOWING THE NICOLA VALLEY COAL-FIELD.

The Vancouver, Victoria & Eastern Railway & Navigation Co. is building a line up the Similkameen River valley from the Boundary region towards the Nicola valley. Rails are laid and trains are running to Keremeos, while grading has advanced to Princeton, 60 miles from Merritt. This road, which has connections with the Great Northern railroad at Spokane, Wash., will give the Hill lines a direct entry into Vancouver from the

east. The route as surveyed passes near the Nicola field, and a connection there is a practical certainty.

Construction of the Canadian Northern railroad has advanced as far westward as Edmonton, Alberta. The line of its location-survey extends from Edmonton across the Rocky mountains through the Yellowhead pass and turns southwest to the Thompson river. It passes either through the Nicola valley or not far from it, and continues down the Fraser to the western terminus at Vancouver. A general election, called for Nov. 25, 1909, will decide the questions of the route to be allowed the Canadian Northern, and the amount and form of government assistance to be granted it.

The Grand Trunk Pacific, Canada's third transcontinental road, with terminus at Prince Rupert, crosses British Columbia several hundred miles north of Nicola valley. A survey is now being run for a branch line, in the neighborhood of Edmonton, to Vancouver, passing through the Nicola valley.

The Nicola coal-field lies wholly in the valleys of the Nicola and Coldwater rivers, which unite at Merritt. The area known to be underlain by coal is about 8 miles long by 4 miles in greatest width. The Quilchena field, lying 8 miles farther east, is similar in many respects, but is not developed. A deposit of fine alluvium covers the floor of Nicola valley and forms a deep rich soil at the surface. The coal-measures lie directly beneath it, and outcrop through the alluvium at only a few points on the edges of the valley. From the present workings and the outcrops it appears that the measures form mainly a synclinal trough surrounded by igneous rocks. Near the contact with the latter, at certain places, the sedimentaries are irregular in dip and strike, perhaps showing lesser folds, but the variations seem to be simple and do not interfere with development or mining.

The coal-measures, which Dr. Dawson and Dr. Ells, of the Canadian Geological Survey, term the Coldwater group, seem to have been deposited in the very early part of the Tertiary, wholly previous to the extensive volcanic activity that occurred in Miocene times and produced the present walls of the valley. At the base of the Coldwater group, where coal is lacking, conglomerate-beds appear; higher up in the series the rocks are mostly alternating sandstones and shales with numerous beds

of coal. The workings are not extensive enough to allow a complete correlation of the measures in different parts of the field, but it is probable that there are at least eight or ten workable seams more than 3 ft. thick.

The first mining of any importance was done near Coal Gully, in the southern part of the field, in December, 1906, by the Nicola Valley Coal & Coke Co. This company has opened up considerable ground and is now putting out about 300 tons per day, which is sold mostly to the Canadian Pacific railway. The Diamond Vale Coal & Iron Mines, Ltd., have carried on development near the center of the valley, on both slopes of the syncline, by means of diamond-drill holes, shafts, and slopes. Approximately 6,000 tons of coal have been produced, but present efforts are being confined to development. The Coal Hill Syndicate recently opened two valuable seams, of which one is 10.5 ft. thick. Incidentally, the latter work has extended the known area of the coal-field west of Coal Gully, the place of original discovery. Diamond-drilling, now being carried on by the Nicola Development Co., Ltd., in which I am interested, is proving that the northern portion of the field is valuable, and is of greater extent than any of the maps have shown.

Room-and-pillar is the usual mining-system. The Nicola Valley Coal & Coke Co. mined, perhaps, two or three thousand tons by long-wall, but the experiment did not prove wholly successful. The same company is installing a compressor and 10 Siskol coal-cutting machines to be used in both narrow work and rooms. The thickest seam developed anywhere in the field at this time is the Nicola Valley, No. 1, which measures 18.5 ft. from floor to roof, including a 2-ft. parting of hard clay. The floor of the seams in general is sandstone, less often hard shale. Less than half the present gangways and slopes are timbered, owing to a dependable roof, which is mainly fine-grained sandstone, sometimes with "following stone" of "slate" from 4 to 8 in. thick. In most places the seams are solid enough to yield a very good proportion of lump in the run-of-mine coal. Dips are encountered as low as 10° and as high as 40°; the usual limits, however, appear to be 15° and 30°. The amount of "gravity" coal in the valley is quite limited. All the workings are dry, except in a few places where the water evidently penetrates from the surface. Traces

of gas have been detected in the past at certain points; recently it has become quite noticeable in one of the pits.

All of the mining- and parts of the hauling- and loading-work are done by contract. For seams less than 5 ft. thick the contract-price of coal on the cars is $1 per ton, while for thicker seams 80 cents is paid. Underground laborers receive average wages of $3 for an 8-hr. day; outside men, $2.75 for 9 hr. Posts and caps of yellow pine from the surrounding hills cost 5 cents per foot. Stulls of cottonwood from the river-bottoms cost 10 cents each.

Nicola Valley coal is bituminous in character, and yields an excellent coke for copper-furnace and blacksmith purposes. The composition varies somewhat in the different seams, but in a given seam it appears to be constant. The moisture is unusually low for a Western coal, running from 2 to 7 per cent. The volatile constituents vary from 32 to 39 per cent., and the fixed carbon from 49 to 57 per cent. The amount of ash in the coal when it reaches market can be kept within a reasonable limit, say from 4 to 8 per cent. Although some of the seams contain bone or shale, which is either sorted out by the miners or picked out at the tipple, the coal itself is clean. In the mining- and handling-operations fine dirt accumulates in the coal, particularly with the small sizes, but the latter respond readily to jigging-tests, and yield products low in ash. Sulphur, in the form of pyrite, varies from 0.5 to 1 per cent. A large lump of Diamond Vale coal, which had been exposed to the air for some time, gave the following analysis:

	Per Cent.
Moisture,	2.66
Volatile constituents,	37.84
Fixed carbon,	55.14
Ash,	4.36
Total,	100.00

Markets for Nicola Valley coal and coke are near at hand. The three railroads already mentioned will have long stretches of road, from 300 to 400 miles each, in which Nicola will have the advantage of short haul in comparison with other fields. In addition to the main lines, branches are certain to be built soon through the nearby mining and agricultural districts of British Columbia and Washington. The mining and smelting industries of southeastern British Columbia, which are increasing in importance, now obtain their coal and coke from distant

points at high cost. Lastly, the demand for coal to be used for domestic supply and for power purposes is growing steadily. Although stock-raising and lumbering will continue to be important industries in the region, agriculture and horticulture have gained firm footholds. Many of the valleys, including the Nicola itself, have been found capable of producing under irrigation abundant crops of the finest grades of apples and other fruits. With the influx of country settlers, naturally the cities and towns are thriving. In general, southern British Columbia is keeping pace with the rapid development of the Pacific Northwestern States.

Postscript (Mar. 16, 1910).—The daily production of coal in the Nicola valley has increased somewhat during the past winter and development-work has continued. The coal appears to be suited to the use of the coal-cutting machines that were installed last fall. The railroad situation has improved materially, largely as a result of the election held in November. The provincial government of British Columbia has given aid to the construction of the Nicola branch of the Canadian Pacific railway southeastward from Nicola through Penticton to Midway, the present terminus of the branch leading from the east. The Vancouver, Victoria & Eastern Railway & Navigation Co. has let contracts to extend the road on the western side of the Cascade mountains from Abbotsford to a point near Hope, and, on the eastern side, from Princeton to Otter Flat, leaving a gap of about 60 miles to be completed across the summit of the range east of Hope. Regarding the Canadian Northern railway, bonds to the amount of $35,000 per mile have been voted for its construction from Vancouver up the Fraser, Thompson, and North Thompson rivers to Yellowhead pass. Right-of-way, town-sites, and timber for construction purposes have been granted free wherever the road traverses crown lands. The southern branch of the Grand Trunk Pacific now seems likely to follow a route considerably west of Nicola valley.

General References: *Report on the Mining and Metallurgical Industries of Canada*, 1907–8, pp. 260 to 271. Canada, Department of Mines, Mines Branch, Ottawa (1909).

Annual Report of the Minister of Mines, British Columbia, for 1905, pp. J196 to J201. Victoria, B. C. See also reports for later years.

Summary Reports of Geological Survey Branch of Department of Mines, for 1904 and later years. Ottawa.

Influence of Top-Lag on the Depth of the Pipe in Steel Ingots.

BY HENRY M. HOWE,* NEW YORK, N. Y.

(Spokane Meeting, September, 1909.)

IN my original paper, Piping and Segregation in Steel Ingots, I pointed out[1] among other things that, in view of the slighter stretching (virtual expansion) of the crust, and greater opportunity for sagging, there should be less piping in broad than in narrow ingots, and less in slowly-cooled ingots, *e.g.*, those cast in pre-heated sand molds, than in those which cool quickly, *e.g.*, those cast in iron molds. A. A. Stevenson[2] said that neither of these predictions agreed with his own experience. In particular, in a picture which he showed of a wide and of a narrow ingot cast from the same ladleful of steel,[3] the wide ingot had certainly piped much more deeply than the narrow one.

At the time I did not see the explanation of these discrepancies, but further reflection makes it evident.

One of the most important elements in determining the depth of the pipe is the degree of "top-lag," that is, the degree to which the solidification of the top of the ingot lags behind that of the bottom. Through this lagging the steel of the upper part of the ingot is able to run down and fill the pipe below as fast as it forms. To the importance of this lagging I called attention in my original paper.[4]

Sand vs. Iron Molds.—If we compare two like ingots, one cast in an iron and the other in a sand mold, we see that the top-lag is much greater in the former than in the latter, because in the former the lower part of the ingot is cooling off fast while the metal is running into the upper part. It is perhaps easier to look at this as a case of the solidification of the bot-

* Professor of Metallurgy, Columbia University, New York, N. Y.

[1] *Trans.*, xxxviii., 56 to 58 (1908). [2] *Idem*, xxxix., 834 to 836 (1909).

[3] *Idem*, 837. [4] *Idem*, xxxviii., 58.

tom outrunning that of the top, which is nothing but the other aspect of top-lag. This stronger top-lag in case of iron than in case of sand molds may well outweigh the influence of greater opportunity for sagging which the sand mold gives. It is to this effect that I refer the discrepancy between Mr. Stevenson's observation and my prediction. The latter ought to have been modified so as to take into account the greater top-lag in the iron mold. If this influence can be cut out, then the effect of greater opportunity for sagging in the sand mold should become evident in the shortening of the pipe. In experiments which I have since tried I have found this to be the case.

A striking example of the shortening effect of slow cooling, which, as I asserted, ought to shorten the pipe, is given in the case of ingots which solidify slowly in the soaking-pit. Their pipe is much shorter than that of ingots which solidify rapidly in the outer air.

Wide vs. Narrow Ingots.—The case which Mr. Stevenson gives, in which a narrow ingot piped much less deeply than a wide one cast from the same ladleful of steel, is seen, on further consideration, not to be a fair contradiction of my prediction, for two reasons. In the first place, the fact that these two ingots were cast with the wide end up tends to shorten the pipe much more in the narrow than in the wide ingot. I have insisted on the effect which having the large end up has of shortening the pipe by means of top-lag,[5] though I had not at that time devised this term. It is clear that the effect of this taper is much greater in a narrow than in a wide ingot. The taper is usually the same, and hence the absolute widening of the top is the same, in narrow as in wide ingots, and hence it forms a much larger proportion of the width of the ingot in narrow than in wide ingots. But the mere fact that the widening at the top bears a greater proportion to the average width of the ingot in narrow than in wide ingots has for its clear result that this widening causes more top-lag in narrow than in wide ingots. The effect of width as such on the depth of the pipe can be shown only when the effect of other variables is cut out. Now in this case the greater top-lag caused by the taper in the narrow than in the wide ingot directly opposed the effect of width as such in permitting sagging and in lessening crust-stretch. In order to test

[5] *Trans.*, xxxviii., 60 (1908).

the effect of width as such, parallel-sided ingots should be used, and the effect of other variables should be excluded. This I have done in certain preliminary experiments, which, as far as they go, support my prediction that width tends to shorten the pipe.

In case the ingots are tapered in the opposite direction, with the large end down, this taper, because it tends to lengthen the pipe, and because the effect of taper should be inversely proportional to the width of the ingot, should tend to lengthen the pipe more in narrow than in wide ingots. In fact, this influence is relatively unimportant in wide ingots.

The second reason why the evidence given by Mr. Stevenson's wide and narrow ingots is not valid is that the narrow ingot was poured much more slowly than the wide one, and this in itself, as I pointed out clearly,[6] has an important effect in shortening the pipe. It was evidently poured much more slowly than the wide ingot, because the two were in the same bottom-cast group, and consequently the steel must have entered the narrow ingot very much more slowly than the wide one.

Everything else being equal, the more-rapid cooling of the bottom of a narrow than of a wide ingot tends to give the former greater top-lag than the latter, and thus to shorten its pipe.

Looking at it in a general way, we see that narrowness in one way tends to shorten the pipe and in other ways tends to lengthen it. On one hand, in that it leads to (1) the more-rapid cooling of the bottom, it increases top-lag and thereby tends to shorten the pipe. On the other hand, narrowness tends to lengthen the pipe (2) by leading to relatively great crust-stretching (virtual expansion), (3) by giving little opportunity for sagging, and (4) (for given rate of pouring) by leading to rapid rise of the surface of the metal, and in this way lessening the top-lag. My original prediction, supported by the observations which I had then made, was based on these latter considerations, (2), (3), and (4), and overlooked consideration (1). Now it may be shown hereafter that my prediction does not hold true under certain comparable conditions, or even under most comparable conditions. But Mr. Stevenson's

---

[6] *Trans.*, xxxviii., 60 (1908).

evidence does not prove this, because if my prediction is true that the net effect of narrowness as such is to lengthen the pipe, nevertheless this effect might be completely masked under his conditions by the joint effect of (1) his having the large end up and (2) his pouring more rapidly into the wide than into the narrow ingot, because both of these things should tend to give the narrow ingot the shorter pipe of the two.

In other words, his conditions introduced certain accidental concomitants of narrowness, which concomitants clearly tend to shorten the pipe in his narrow ingots, and thus to mask the influence of narrowness as such. The fact that in the presence of these pipe-shortening concomitants the predicted pipe-lengthening effect of narrowness as such is not seen, is no proof either that that effect does not exist or that it would not be seen when such masking concomitants are absent.

I hope to present further data on this subject soon.

The Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes.

POSTSCRIPT.

BY H. O. HOFMAN AND W. MOSTOWITSCH, BOSTON, MASS.

(Spokane Meeting, September, 1909.)

In our investigation of the Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes,[1] we incidentally studied the decomposition of ferric oxide when heated in a current of dry air. We found that at 1,500° C., with 15-min. periods of heating, Fe_2O_3 hardly changed in weight, the greatest loss being 0.23 per cent., and we therefore concluded that the compound remained unchanged at 1,500° C. in a current of dry air. In a more recent experiment we heated Fe_2O_3 in a current of dry air and found that there was a decided loss in weight. A new investigation upon the behavior of Fe_2O_3, using a larger amount of substance and heating for a longer period of time, revealed the fact that a measurable dissocia-

---

[1] *Trans.*, xxxix., 628 to 653 (1909).

tion took place at 1,375° C. The experimental data are given in Table I.

TABLE I.—*Decomposition of Ferric Oxide by Heating in Dry Air.*

Sample No.	Weight of Substance After Heating at 1,000° C.	Oxygen in Substance.	Temperature.	Time.	Loss in Oxygen.		Remarks.
	Grams.	Grams.	Deg. C.	Min.	Mg.	Per Ct.	
12	0.4584	0.1377	1,000	15	none.	none.	No change.
	0.4584	0.1377	1,100	15	none.	none.	No change.
	0.4584	0.1377	1,200	15	none.	none.	No change.
	0.4584	0.1377	1,300	15	0.6	0.43	Sintered, not magnetic.
	0.4584	0.1377	1,400	15	7.6	5.52	Sintered, magnetic.
13	0.4438	0.1333	1,100	40	none.	none.	No change.
	0.4438	0.1333	1,200	15	none.	none.	No change.
	0.4438	0.1333	1,300	30	none.	none.	Sintered, not magnetic.
	0.4438	0.1333	1,400	15	6.6	5.0	Sintered, magnetic.
	0.4438	0.1333	1,400	15	7.2	5.4	Sintered, magnetic.
	0.4438	0.1333	1,400	15	8.2	6.2	Sintered, magnetic.
14	0.3866	0.11615	1,100	25	none.	none.	No change.
	0.3866	0.11615	1,350	15	0.4	0.34	Sintered, not magnetic.
	0.3866	0.11615	1,350	15	0.2	0.17	Sintered, not magnetic.
15	0.2738	0.08226	1,100	20	none.	none.	No change.
	0.2738	0.08226	1,360	30	none.	none.	Sintered, not magnetic.
	0.2738	0.08226	1,375	15	3.6	4.4	Sintered, magnetic.
	0.2738	0.08226	1,375	15	4.2	5.1	Sintered, magnetic.

The results in Table I. show that Fe_2O_3 heated to 1,360° C. in a current of dry air at atmospheric pressure for 30 min. remains chemically unchanged, and that at 1,375° C. the loss in oxygen amounts to 4.4 per cent. The substance which at 1,360° C. has sintered but was not attracted by the magnet, becomes magnetic at 1,375° C. and contains some FeO.

The results of our new experiments agree with those of P. T. Walden,[2] who upon heating Fe_2O_3 in an evacuated glass tube found that the pressure of the oxygen liberated rose from 5 mm. at 1,100° C. to 166 mm. at 1,350° C. Since this pressure corresponds to about one-fifth of an atmosphere, or to the partial pressure of the oxygen in the air, he concluded that Fe_2O_3 was stable in air to approximately 1,350° C.

[2] *Journal of the American Chemical Society*, vol. xxx., No. 9, pp. 1350 to 1355 (Sept., 1908).

The Formation and Enrichment of Ore-Bearing Veins.

SUPPLEMENTARY PAPER.

BY GEORGE J. BANCROFT, DENVER, COLO.

(Spokane Meeting, September, 1909.)

At the New York meeting of the Institute, April, 1907, I presented a paper entitled, The Formation and Enrichment of Ore-Bearing Veins,[1] in which paper I advanced the following propositions:

(1) That the majority of mineralized veins are the product of expiring vulcanism; (2) that most of these veins were primarily mineralized by comparatively rich solutions in comparatively short periods of time; (3) that the solutions gained their metal-values from a comparatively rich source; (4) that there is a barysphere containing large amounts of the useful metals; (5) that eruptions spring from various depths and bring various kinds of magmas towards the surface; and (6) that only those eruptions which disturb the barysphere and bring a magma rich in metals sufficiently near the surface to be leached by vein-making solutions are productive of valuable ore-deposits, other eruptions producing barren veins. Ore-deposits due to magmatic segregation were not included in this general survey.

As a result of considerable further study I have modified my views in some respects, while in others I feel more sure than ever of the ground taken at that time.

That ore-bodies are infrequent occurrences and born of extraordinary conditions I think is now generally accepted. The theory that persisted for a time—namely, that ore-bodies were formed by the ordinary ground-water, which consists of extremely dilute solutions, derived from leaching extremely lean surface-rocks, and which must occupy enormously long periods of time in concentrating the values so leached, is, I think, now pretty generally regarded as not applicable to the great majority

---

[1] *Trans.*, xxxviii., 245 (1908).

of our mining-districts, although it may account for a few isolated deposits.

I note that a few recent writers still use this theory as a sort of "point of departure" for their discussions, but the majority seem to realize that the tendency of ordinary ground-water circulation is to diffuse any soluble matter rather than to concentrate it, and that an unusual precipitant must be present to provoke an important concentration under this hypothesis.

That the forces of expiring vulcanism are the agencies which account most logically for ore-bodies is an opinion very generally held, not only because of the intimate association of ore-bodies with eruptive rocks, but also because the study of active volcanoes and of the springs rising near them shows that ore-making agencies to a limited extent, at least, are at work there. Thus, A. Lacroix[2] found pyrite, pyrrhotite, and galena, together with sulphates of sodium, potassium, calcium, magnesium, and aluminum, in the sublimates of fumaroles on Vesuvius; J. W. Mallet[3] discovered silver in volcanic ash at Cotopaxi and Tunguragua; O. Silvestri[4] found copper in the fumes of Etna, while traces of practically all the common metals have been found in eruptive rocks.

Similarly, it has been recognized that the period of expiring vulcanism could not have been of very long duration, geologically speaking, although it has been shown that in some districts the repeated recurrence of eruptive action has had the effect of continuing the mineralizing action through long periods of time.

My fifth and sixth propositions have not been so well established. I think it is generally admitted that certain eruptive rocks produce mineralized areas, while others do not; this suggests most forcibly that the eruptives themselves differed very decidedly in the matter of mineral content; and it seems reasonable to infer that those eruptives which do produce mineral richness have been rich themselves and probably originated at great depth. The advance of science has still further substan-

[2] *Bulletin de la Société Française de Minéralogie*, vol. xxx., p. 219 (1907).

[3] *Proceedings of the Royal Society*, vol. xlii., p. 1 (1887); vol. xlvii., p. 277 (1889–90).

[4] *I Fenomeni Vulcanici presentati dell' Etna, etc.* (Catania, 1867), per *Bulletin* No. 330, *U. S. Geological Survey*, p. 217 (1908).

tiated the theory that the earth has a very heavy center, and it is reasonable to suppose that this increased specific gravity is partly due to relatively large metal-content. It is not, however, generally conceded that it is necessary for a mineralizing eruptive to be so rich in metals and so heavy that it would rarely or never reach the surface, but would form laccolites, according to my old hypothesis. Nor do I longer hold this view. At that time I remarked:

> "If it could be shown that the surface eruptive rocks have a tendency to throw off metals, as they do steam and sulphur, during the cooling process this would remove many of my objections to considering them the source of the metals in our ore-bodies. In the lack of such proof, however, we must recognize that they are extremely lean, and therefore a very unlikely source of mineral wealth."

Now, I think there are very good reasons for believing that this very thing is true—namely, that eruptive rocks do have a tendency to throw off their metal-content during the cooling process, or, rather, as soon as they reach a horizon of lessened pressure, which condition is apt to be coincident with cooling.

The wonderful crystallographic researches of J. E. Spurr, Waldemar Lindgren, and others, have shown that magmas are totally different from dry melts, and the cooling of a magma is accompanied by a remarkable series of differentiations. Mr. Spurr has shown how the metals would be concentrated either in a very base or a very acid magma, and finally, in the case of the acid magma, would be extruded together with pure silica and water, thereby forming veins. I think, however, that those who have made a great deal of the powerful agency of magmatic differentiation have overlooked a very active contemporary agent—namely, chlorine. Bromine, iodine, and fluorine may be equally active agencies in volcanic emanations; but as these elements are relatively rare, I shall confine myself here to the very active part which chlorine may play in carrying away from the hot eruptive its metal-content and depositing the same in ore-bodies. These considerations have been suggested to me by studying the dry-chlorination process for the treatment of complex ores, as developed by J. L. Malm, at Corbin, Mont., and at Denver, Colo. This process depends primarily upon the facts that, in the dry state, chlorine has a greater affinity for the metals than sulphur or oxygen, and that the chlorides of all the metals are soluble together in hot water.

Thus, cupric chloride, lead chloride, zinc chloride, gold chloride, and iron chloride are soluble in hot water direct, and silver chloride is soluble in hot cupric chloride.

Now, it is noticeable that chlorine is nearly always present in volcanic emanations. Thus, J. W. Judd says:[5]

> "The most abundant of the substances which are ejected from volcanoes is steam or water-gas, which, as we have seen, issues in prodigious quantities during every eruption. But with the steam a great number of other volatile materials frequently make their appearance. The chief among these are the acid gases known as hydrochloric acid, sulphurous acid, sulphuretted hydrogen, carbonic acid, and boracic acid; and with these acid gases there issue hydrogen, nitrogen, ammonia, the volatile metals arsenic, antimony, and mercury, and some other substances."

After dwelling upon the large amount of CO_2 present in volcanic gases, Chamberlin and Salisbury have the following to say with reference to emanations:[6]

> "Sulphur gases are very common accompaniments of volcanic eruptions. They take the forms of sulphuretted hydrogen and sulphurous acid and perhaps of sublimated sulphur, all of which are liable to pass by oxidation and hydration into sulphuric acid. Chlorine and hydrochloric gases are also common, particularly at high temperatures. Fluorine and other gases are occasionally present."

T. Wolf found that near the crater of Cotopaxi the fumes were mostly of hydrochloric acid with some free chlorine, while at lower levels hydrogen sulphide was found with a trace of sulphur dioxide.[7]

Sainte-Claire Deville found chlorides of iron and copper in a fumarole of Vesuvius.[8]

R. Bunsen found various metallic chlorides in the sublimates around fumaroles of Mt. Hecla in Iceland.[9]

A characteristic of volcanic emanations is, that chlorine is found either free or as the chloride of the metals, or as hydrochloric acid, while a characteristic of hot mineral springs is that the water contains large quantities of the chlorides of sodium and potassium. Granting that such hot mineral springs

---

[5] *Volcanoes: What They Are and What They Teach*, p. 40 (1881).

[6] *Geology*, 2d ed., vol. i., p. 619 (1906).

[7] *Neues Jahrbuch für Mineralogie, Geologie und Palaeontologie*, p. 163 (1878).

[8] *Bulletin de la Société Géologique de France*, Second Series, vol. xiii., p. 606 (1855–56).

[9] *Annales de Chimie et de Physique*, Third Series, vol. xxxviii., p. 215 (1853).

are related to eruptions, and that the chlorine in both emanations and springs had a common origin, it is evident that the part which has escaped by the medium of spring-water has undergone certain reactions, which the part which escaped as a hot gas, from an open vent, did not undergo. Does not this suggest what may take place in case the chlorides are extruded through crevices or veins where they may encounter gradually lessened temperatures together with water?

Now, it is known that the temperature of fluid magmas may range up to 3,000° F.[10]

Let us suppose that, under a temperature of 1,100° C., we had a magma which contained chlorine, water-gas, sulphur, silver, copper, lead, iron, zinc, and gold. Of course, there would also be other elements present, but, as previously stated, I shall not try to cover the whole field. Let us suppose that there is sufficient chlorine to form chlorides with all the metals and an excess besides. Any hydrogen present would be combined with the chlorine, the affinity for chlorine being greater than for sulphur. Now, let us try to imagine what would happen as this magma approached the surface.

At the temperature given, the chlorine would attack all the metals, except gold, and the chlorides would all be gases, for it is well known that the metallic chlorides are all volatile at relatively low temperatures.

The chloride of gold under atmospheric conditions decomposes at about 120° C. Of course, the volcanic gases are under some pressure even when escaping from the magma, and, as pressure raises the temperature of decomposition, it is difficult to say just how cool the magma must become before the gold would accompany the other metals.

The fact that gold is often found by itself in a state of great purity may be accounted for by its isolation from the other metals as regards chemical reactions. Thus, at Farcum Hill, Breckenridge, are found deposits of most beautiful crystalline gold by itself, while less than a mile away are large deposits of complex ores. The complex ores occur in the eruptive dikes or on the contacts, while the gold is found in carbonaceous shale. It is well known that carbon will precipitate gold from

---

[10] *Geology*, Chamberlin and Salisbury, 2d ed., vol. i., p. 615 (1906).

a chloride solution, as is done in wet-chlorination mills, but it will not precipitate the other chlorides.

The chlorine would not attack the silicates of the eruptive, as is shown by the experiments of Brun,[11] who heated a Lipari lava and observed the following exhalations:

From 0° to 825°, volatilization of water.

At 825°, first evolution of chloride vapors.

From 874° to 1,100°, temperature of explosions.

At 1,100°, mean temperature of flowing lava.

Although the dry-chlorination process does not use high temperatures, Mr. Malm has experimented with chlorine and the ordinary rock-minerals at high temperatures and has shown that they do not react.

The sulphur would occur as sulphur gas and partly as sulphur chloride. There would, of course, be a great deal of water-gas present. As the magma rose to a horizon of lessened pressure these gases would expand and leave the magma, bursting out through any veins or vents or porous strata that might provide a means of escape. The great distance to which the metals have penetrated porous strata, as, for instance, at Morenci, Ariz., may be accounted for in this way.

The farther from the magma the gases traveled the cooler they would become, and as they became cooler they would become a liquid. We would then have the chlorides of the metals in a hot aqueous solution, together with sulphur chloride and elemental sulphur, the latter in an extremely fine state of subdivision (as it is found in many mineral springs).

Precipitation would not take place till some precipitating-agent was encountered in the rocks.

The cooling of the solution would throw down the lead and the lead only. Silver chloride is difficultly soluble and easily precipitated. This may account for the exceeding purity of some lead-deposits and the frequent association of silver and lead.

If CaO in abundance were encountered all the metals would be precipitated together. This may account for the complex nature of many deposits in porphyry-lime contacts, as, for instance, at Leadville, Rico, Breckenridge, etc.

---

[11] *Archives des Sciences Physiques et Naturelles*, Fourth Series, vol. xix., pp. 439, 589 (1905).

It is fair to assume that the heated eruptive would drive off the CO_2 from any limestone in immediate contact with it. The chloride gases emanating from the same eruptive would find an immediate and abundant precipitant in the form of CaO thus formed, or, in case conditions became cool enough for water to convert the oxide to the hydroxide, the latter would be equally efficient as a precipitant. Thus,

$$2\ MCl_x + xCaO = M_2O_x + xCaCl_2$$
$$2\ MCl_x + xCaO + xH_2O = 2\ M\ (OH)_x + xCaCl_2.$$

In granitic or eruptive rocks, the methods of precipitation are not quite so simple. I have immersed pieces of Boulder County granite in solutions of the metallic chlorides, sulphur chloride, and a little free acid, and at the end of three days there was no appreciable result. Again, I have boiled pieces of the same granite in strong hydrochloric and sulphuric acid, and in half an hour the reactions were very considerable. The hydrochloric acid was the more active. The piece of granite was eaten away considerably. Examination of the remaining piece with a magnifying-glass showed that the surface was reduced to a covering of spongy silica and white mica. Whether the mica had been whitened by leaching the iron, or whether only those bits remained intact which had no iron, I am not prepared to say. In addition to the piece remaining undissolved (at the end of 30 min.), there was a sediment which, on examination, appeared to be rounded and porous pieces of silica and minute flakes of pure white mica. The test with sulphuric acid rendered the piece darker. Only a very small part went into solution. An examination with a glass has shown the feldspars to be lacking and the surface to be covered with a coating of pearly silica and black mica. In this connection Clarke says:[12]

"Hot waters, charged with sulphuric or hydrochloric acid, attack nearly all eruptive rocks, dissolve nearly all bases, and leave behind, in many cases, mere skeletons of silica."

On this subject Judd says:[13]

---

[12] *Bulletin No.* 330, *U. S. Geological Survey*, p. 408 (1908).
[13] *Volcanoes: What They Are and What They Teach*, p. 41 (1881).

"In many volcanoes the constant passage through the rocks of the various acid gases has caused nearly the whole of the iron, lime, and alkaline materials of the rocks to be converted into soluble compounds known as sulphates, chlorides, carbonates, and borates; and, on the removal of these by the rain, there remains a white, powdery substance, resembling chalk in outward appearance, but composed of almost pure silica. There are certain cases in which travelers have visited volcanic islands where chemical action of this kind has gone on to such an extent, that they have been led to describe the islands as composed entirely of chalk."

F. Henrich shares the same view, claiming that the chlorides of potassium and sodium found in sublimates of aqueous fumaroles were formed by the action of moisture and hydrochloric acid on the alkaline silicates of the heated lavas.[14]

We know from observation underground that granitic and other rocks are attacked by mineral solutions which produce first a softening of the rocks, and eventually remove all the original constituents except the silica and hydrous alumina silicates.

It is, of course, difficult to reproduce in the laboratory with a few simple salts the complex reactions that take place far underground; and here at least we must take the evidence as we find it, even if it is difficult to reproduce the reactions.

We find chlorine or hydrochloric acid or metallic chlorides issuing from volcanoes, and we find sodium, calcium, and potassium chlorides issuing from mineral springs; so it seems highly probable that the sodium and potassium silicates break up, and the chlorine, after it has become cooled below the boiling-point of water, reacts with the sodium or potassium, dropping any metals with which it may have been combined.

The reactions leading to the formation of calcium chloride have been explained above. The reactions with the silicates would most naturally begin with the free hydrochloric acid, thereby liberating hydrogen, which would combine with the sulphur to form H_2S, and this in turn would precipitate the metals according to the following reactions:

$$MCl_x + \tfrac{x}{2}H_2S = xHCl + MS_{\frac{x}{2}},$$

and would convert any metals already precipitated by other agencies to the sulphides, as follows:

[14] *Zeitschrift für angewandte Chemie*, vol. xix., No. 30, p. 1326 (July 27, 1906); vol. xx., No. 5, p. 179 (Feb. 1, 1907).

$$MO_x + xH_2S = MS_x + xH_2O$$
$$M + xH_2S = MS_x + xH_2.$$

The first reaction would result in more hydrochloric acid, which would be available to repeat the cycle.

Whether the metals are precipitated by CaO or by reactions with the silicates, they would be almost at once converted to sulphides, according to the above reactions; and that is the condition in which we find them. Gold, which is precipitated by carbon, would not be subject to the action of H_2S, and hence would likely occur as free gold in carbonaceous formations, which agrees with the facts as we find them.

As stated above, I have confined myself, for the sake of simplicity, to an arbitrarily-chosen condition. It is my purpose to point out the very active part which the halogens may play in divesting an eruptive of its metal-contents, and conveying the same into neighboring veins or openings in the rocks, rather than to prescribe the exact steps that are followed. It is conceivable that under certain conditions the above-mentioned reactions may be very important, while in others they may be negligible. It is conceivable that in some cases an eruptive may be totally divested of its useful metals and yet little of it be precipitated short of the atmosphere. This may account for the Snake river placer gold, which is extremely fine, and so often found associated with volcanic ash as to provoke the theory that it had an atmospheric origin. On the other hand, a metalliferous eruptive may be only partly relieved of its metal-content, the residue remaining in a more or less segregated form in the body of the eruptive, as at Ely, Nev., where the sedimentaries adjoining the eruptive area contain veins and deposits of copper-ore, while the eruptive itself contains large masses of low-grade ore which appear to be due to magmatic segregation.

I think these considerations, helping to explain some of the puzzling things about ore-bodies, may be of service in promoting progress towards the goal of a perfect understanding of the subject.

DISCUSSIONS.

Piping and Segregation in Steel Ingots.

Discussion of the paper of Henry M. Howe, *Trans.*, xxxviii., 3, 924; xxxix., 818.

P. H. DUDLEY, New York, N. Y. (communication to the Secretary*):—This renewed discussion of Professor Howe's paper is partly due to a recent conversation with him, in which I called attention to some features of modern practice he had not stated, and he asked me to state them for publication, that eventually the truth might be ascertained. His invitation to present observations and their interpretations, as applied in practice, is accepted in the broad spirit in which it was extended.

My observation during many years of practice in teeming steel ingots has been that piping or shrinkage-cavities and segregation are greater in the higher-carbon steels than in the medium and mild steels. Hence we are obliged to discard a larger portion of the ingot for sound high-carbon metal, especially as its dimensions are increased. The necessity of teeming all kinds and grades of steel involves the question of the greatest percentage of sound or available metal free from pipes and sponginess, whether of crucible, Bessemer, open-hearth, or electric manufacture, which can be used for the purposes intended. This question requires renewed investigation, in order that we may secure a better and higher grade of steel, as the metal is subjected to more severe service in the rapid progress of the industrial arts.

Many members of the Institute can recall the rapid failure of the iron rails from 1860 to 1865, when the driving-wheel loads of the engines reached from 10,000 to 12,000 lb. The physical properties of wrought-iron, with its 1 to 1.5 per cent. of included cinder, had been sufficient for the evolution of the railroads, but proved inadequate for their subsequent development. The substitution in 1865 of light Bessemer-steel sections for iron rails was facilitated by the fact that the steel, having been

---

* Received Sept. 10, 1909.

once molten in manufacture, would be more homogeneous than iron.

All of the early Bessemer rails were rolled from ingots which, after being teemed and "stripped in the pits," were allowed to cool for examination, as in crucible practice, to see whether they were suitable for rails. The ingots selected were reheated and hammered into blooms; and many cases of genuine piped rails with smooth unwelded walls of the web, from cold-shrinkage cavities, were subsequently found in the tracks.

In the winter of 1876–77, I was sent to investigate a number of piped rails which had split in the web and broken in the track. In a few cases I found slag between the walls, but in most cases the walls were oxidized. I went also to the mill in which the rails were made and found ingots which had been allowed to cool, of which 80 per cent. were piped at one or both ends. This fact, in connection with some forcible language by the general manager of the mill, in answer to my question, "whether this was an extraordinary percentage, or the usual amount of piped blooms," made a lasting impression on my mind. I concluded that the cavities represented the interior volume of shrinkage from the molten metal of teeming to the cold metal of the ingot-walls, and that the greater part of it could be avoided in rail-ingots by charging them into the reheating-furnace as soon as possible after they were stripped. This method soon became the regular practice at all rail-mills, and the percentage of piped rails from shrinkage-cavities was reduced. Professor Howe does not give this feature of the "state of the art."

I have always worked upon the plan of checking the full amount of the interior shrinkage of the volume of the steel in the ingots for rails by prompt reheating after stripping, in connection with the chemical composition required for the physical properties and degree of deoxidation desired for the grade of steel made.

The adjustment of the chemical composition, first, to secure the proper physical properties, and, secondly, to obtain as sound ingots as practicable under the existing conditions of manufacture, was coincident with my design and use of heavy sections. I did not expect good results in manufacture unless I made proper provision to secure them; and the freedom from piped

rails in more than 500,000 tons, which I made from 1891 to 1897 inclusive, is evidence to me that the theory upon which I worked had basis of fact. I can now control my chemical composition for the section and size of ingot, and with proper time for the teeming, stripping, and charging into the reheating-furnaces, can secure practically pipeless rails. My statements refer to rail-ingots, and to secure the desired results my specifications are not universal, but have always had an adaptability to meet local conditions of manufacture at different mills.

In teeming ingots for basic open-hearth rails, the steel does not, as a rule, set "dead," as it does in Bessemer practice, but the escaping gases eject sparks and the metal rises in the molds. A cast-iron plate is often placed on the top of the steel, and in some mills the top of the mold is capped and keyed. Aluminum is often used on top of the metal in the molds to quiet basic open-hearth steel for plates as well as for rails. Bessemer steel of from 0.10 to 0.30 per cent. of carbon for billets, splice-bars, and bolts is often teemed in bottle-mouthed molds which are not completely filled, but are capped and keyed to prevent the rising metal from overflowing the molds before it freezes. In teeming the ingots of the various kinds and grades of steel there are distinctive methods of practice, which have been evolved after years of experience in producing the several grades of steel required in service.

Professor Howe, studying the shrinkage-cavities or pipes from cold ingots in which the entire shrinkage of the metal from the teeming-temperature to that of cold steel has occurred, of course finds conditions of manufacture which require attention and correction. Operating-men at the mills and engineers have also studied the obstacles to be overcome, and applied remedies with such success that the conditions are decidedly better than they would seem to be from a study of the cold ingots. There is still much to be done in the way of improvement and progress, which will continue as long as steel is made and teemed into ingots. A method has not yet been discovered by which an ingot can be teemed, its non-piping period checked or prolonged at one stage of the process, and then cooled so as to retain what had been gained, as though the entire manufacture, as far as the metal was concerned, had been completed before the steel cooled. A good estimate of the effectiveness of a

method of checking the shrinkage-volume can be made thus: Take an ingot from a heat, allow it to cool, and then cut and measure its cavity; then take a bloom-crop, the discard from an ingot of the same heat, and cut it. In good practice the shrinkage-cavity will be but a small percentage of that found in the cold ingot, and will indicate the reduction secured.

Fig. 1 is a photograph of a three-rail ingot, for 100-lb. rails, teemed in a mold 19 in. square on the base, 17 in. square on the top, and 66 in. long. The ingot, poured 50.5 in. long, was well deoxidized, and therefore had a large cavity. The ingot had a volume of 7.4 cu. ft., inclosing a shrinkage-cavity of about 128 cu. in., practically 1 per cent. of its volume. This is a larger percentage than would be found in rail-steel not so well deoxidized, or which contained numerous blow-holes.

Fig. 2 is a photograph of the bloom of an ingot of the same heat and length, cut for the 9-per cent. mill-discard. The ingot, after stripping and a subsequent ride of 500 ft., was charged directly into the reheating-furnace without allowing the temperature to fall below the recalescence-point, while the bulk of the steel was several hundred degrees above, and in about 2 hr. the ingot was drawn and bloomed. The cavity was small and less than one-tenth of that of the cold ingot of the same heat. I have had a number of ingots and crops cut in recent years, since large planers have been available in the machine-shops. In former years I was obliged to rely upon ingots broken as "stickers" at the drop for a view of the shrinkage-cavities in those for rails. Professor Howe did not mention this advance in practice from the early days of Bessemer, nor have I seen it reported by other authorities.

Fig. 3 is a photograph of an ingot teemed in a similar mold, but poured 53.5 in. long for a 19-per cent. discard, and of a still greater degree of deoxidation than the steel in the ingot shown in Fig. 1. The ingot was slightly inclined when the photograph was taken, and the top does not show apparently as large as in Fig. 1, but it is of practically the same size. The shrinkage-cavity amounted to 250 cu. in., or 1.4 per cent. of the volume of the ingot. The cavity is not bell-shaped or with sides showing parabolic curves, but nearly vertical, by reason of the jolting in the ride to the stripping-machines, then to the scales to be weighed, then to the reheating-furnaces

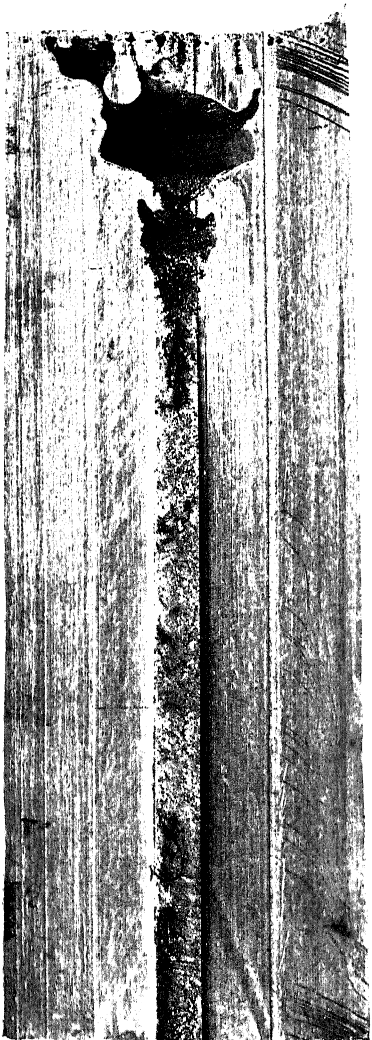


FIG. 1.—SECTION OF INGOT, 17 IN. SQUARE AT TOP, 19 IN. SQUARE AT BASE, AND 50.5 IN. LONG, CONTAINING CAVITY OF 128 CUBIC INCHES.

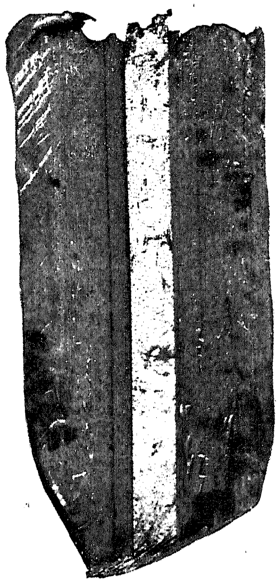


FIG. 2.—BLOOM FROM AN INGOT OF SAME HEAT AND OF SAME SIZE AS FIG. 1, SHOWING REDUCTION OF CAVITY.

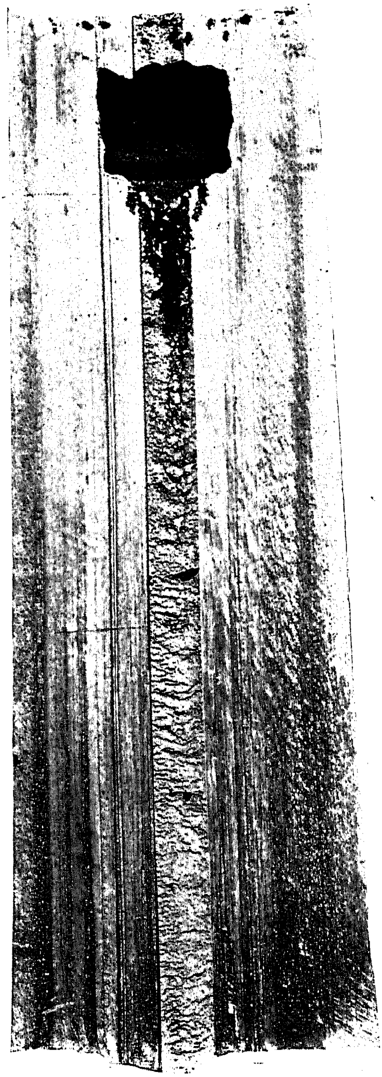


Fig. 3.—Ingot of Same Section as Fig. 1, and 53.5 in. Long, Containing Cavity of 250 Cubic Inches.

Fig. 4.—Bloom from an Ingot Similar to Fig. 3.

holding the other ingots, then 1,000 ft. further to the gantry-crane, where it was cooled. It is important to observe that even after its arrival at the gantry the interior was molten under the bottom of the cup-shaped cavity, and a decided shrinkage of the metal occurred below, the walls being lined with pine-tree crystals. The full length of the 19-per cent. bloom-crop could not be cut at one shearing but required two cuts; therefore, a full bloom-crop was not obtained for cutting and subsequent photographing. The cavity was slightly larger than that shown in Fig. 2, though in the bloom it was completely removed in the percentage of discard, and less discard would have answered.

Fig. 4 is from a crop which, by reason of greater deoxidation, is quite similar to one cut from an ingot having the full percentage increased cavity.

The freezing of the steel for my rails commences on the stool and sides of the molds after a few seconds of contact, and bridges over the top of the ingot in 2 or 3 min. after pouring ceases. Interesting phenomena occur in from 3.5 to 4 min. from the end of pouring, through the expansion of the molds from the sides of the ingot of $\frac{1}{32}$ to $\frac{1}{16}$ of an inch. This expansion of the molds continues until the ingots are stripped in the usual time of manufacture. The exterior of the molds, which measure from 27.5 to 29.5 in. over all, will increase after teeming from $\frac{1}{4}$ to $\frac{3}{8}$ in. per side.

The deoxidizers used seem to make a difference in the quick freezing of the steel, and must be understood and adapted to the practice of the mill in checking the shrinkage-cavities. Aluminum thrown into the molds during teeming, even in small percentages, will often cause the ingot to pipe deeply, while in other cases the pipe will be separated by solid metal into two or more parts. Some of the badly-piped rails which developed in the track during the past two or three years, after about that period of service, were those in which the manufacturers had used aluminum to make "wild" steel set "dead" in the molds without reducing the amount of the other deoxidizers specified by the purchaser. It gives a quicker-setting steel of greater viscosity and apparent shrinkage, which must be stripped and charged into the reheating-furnaces in less time after teeming than the same grade of steel without the alumi-

num, in order to avoid large shrinkage-cavities. In a number of experimental tests made to study the increased shrinkage, it was only with extreme care that it was possible to avoid piping in ingot-molds of the size mentioned, when poured 61 in. long for four 33-ft. lengths of 100-lb. rails. I was then dealing with from 1.2 to 1.5 per cent. interior shrinkage of the volume of the steel, had the ingot been permitted completely to cool. The time of the non-shrinkage- or piping-period was so reduced that it was not sufficient to avoid piping some of the "A" or top rails of the ingots by the usual conditions of practice, which were effectual in holding under sufficient control the shrinkage of ingots for three 33-ft. lengths of 100-lb. rails in the same molds.

The special tests mentioned were made to study the distinction between true piped rails and "split-heads," reported by the trackmen as piped rails, and not as easily detected in the manufacture. My investigations show, I think, that the majority of the "split-heads" with which I have had to deal in the track have occurred in the central core of segregated metal in the heads and webs of the rails. The central core is capped in the bearing-surface by a layer of metal rolled from the exterior of the ingot and often containing slag inclusions. The heads of these rails as manufactured are solid and cut solid by the saws, the inspectors at the mills pass them as sound rails, which they are to all ordinary observation. Laid in the track the metal in the bearing-surface of one or more portions in the length of the rail is inadequate to sustain and distribute the wheel-loads without spreading. Although this takes place only to an infinitesimal amount for each passing wheel, the increment is cumulative, and the rail-head is deformed and eventually splits, after one or more years' service. Etching the top crop would indicate segregation, and often, but not always, shows streaks of cast-iron cut out of the stool by the hot stream of metal in teeming the ingot. This infusion of cast-iron I have found in most of the split-heads which have been investigated in full detail, and is the disturbing factor in many cases of decided segregation. I have found the streaks in a 0.50-per cent. carbon rail to range as high as from 0.80 to 1.02 per cent. of carbon by combustion, though from 10 to 15 points lower by the color-test. The absence or presence in rail-steel

of carbon streaks from cast-iron cut out from the stools or molds has been a disturbing element in studying segregation from the analysis of drillings from different parts of the ingot. It is not uncommon to see in a double stool of an ingot-car that under one mold only a trace of cast-iron has been burned from the stool, while from 30 to 40 lb. has been removed from the other and distributed in the steel of some of the ingots.

The decided segregations when they form part of the head with the cast-iron infusions are often unable to sustain the traffic until the rail is removed for wear, but fail by detailed fractures as a split-head, as already described. When the segregated portion forms the lower part of the web and base, breakages or detailed fractures of the rail as a girder often occur.

I have related briefly some of my observations and practice of many years' experience in the manufacture of heavy and stiff sections of rails. Since the publication of Professor Howe's paper I have tried by measurements to verify his theory of virtual expansion of the walls of the ingot. Professor Howe in his paper says: "If, for instance, on reaching a temperature of 1,000° C. the virtual expansion were such that the ingot was 1 in. wider," etc. I did not expect in rail-ingots to find an increase of any such amount, which for the ingots 19 in. square on the base would augment them to 20 in. square, an increase of 39 cu. in. for each inch in length of the ingot, and for those 50 in. in length, of 1,950 cu. in., a volume from 8 to 10 times greater than the cavities found in the ingots cut, which were well deoxidized. Calipering hot molds and ingots as soon as stripped was attended with so many variations of temperature of the molds and ingots that approximate measurements did not definitely indicate an expansion of the ingot-walls. Though all of the molds were made from the same drawings, and were alike for manufacturing purposes, yet each was of a different size, and each was again modified by its temperature when the ingots were teemed. The cold ingots from the same class of molds also varied as to precise size.

In open-hearth ingots showing ejections of sparks and boiling of the steel against the sides of the molds, the metal often rises 3 or 4 in. and "reams in" before it sets on top. This is a virtual rising of the metal on the top of the ingot, producing

spongy steel, and sometimes a cavity is formed when the ingots cannot be charged promptly after stripping. The contraction of solid steel above the critical points is at a faster rate than below. To cut a 33-ft. 100-lb. rail, the saws are set (for rolling at 1,000° C.) at 33 ft. 6.75 in. Calculate the contraction from figures obtained below the critical points, and it is only about one-half of the above amount. The roll-designer in making a hot template allows a contraction of $\frac{3}{16}$ in. per foot.

Molten to frozen steel, when well deoxidized, seems to have a still higher rate of shrinkage as affecting the respective volumes, and it is important to control the temperature-lag as much as possible in teeming and reducing ingots to solid merchantable forms. There is much which can be effectively done in a practical way. During the past year one steel company, for its rail-ingots as soon as teemed through a 1.5-in. nozzle, throws on the top of the steel a shovelful of from 3.5 to 4 lb. of coke-dust, which ignites, keeps the molten steel fluid in the center of the top of the ingot, and feeds the shrinkage-cavity. Watching the tops of those ingots for 10 or 12 min., I was never able to see a sudden lowering of the molten steel, as would occur in case of a rapid virtual expansion of the walls of the ingot. Those ingots were stripped in from 30 to 35 min. after teeming.

More attention has been paid to producing better ingots for rails in the past two years than previously, in the great demand for quantity. The changes in rail-sections by the railroads did not improve the quality or blooming of the ingots. Sink-heads have been tried experimentally and the shrinkage-cavity reduced. Bottom-pouring for open-hearth ingots has been introduced with decided success, and is promised for rail-ingots in a short time.

The principle of making Bessemer steel for quality instead of for quantity was required for all 1908–09 rails on the New York Central Lines—a return to former practice, the beneficial results of which are already apparent.

Genesis of the Lake Valley, New Mexico, Silver-Deposits.

Discussion of the paper of Charles R. Keyes, *Trans.*, xxxix., 139, 850.

WILLIAM M. COURTIS, Detroit, Mich. (communication to the Secretary*):—I have a few items to add to the history of the Lake Valley mines.

In December, 1879, I was sent to the Bassic mine of Colorado and then to Lone mountain, near Silver City, N. M., and to various copper-camps in Arizona. I staged from Glorietta, N. M., to Tucson, Ariz., stopping at Silver City to see the Cosette mine. This mine led to the opening of the Lake Valley mines, and is a deposit of similar formation.

New Year's day, 1880, found me crossing the Rio Grande above the Jornado del Muerto. At Aleman, Victorio's band crossed four hours ahead of us. We picked up four wounded soldiers to take to Fort Bayard, for during the day the colored troops under General Hatch had been driving the Indians back through the San Andreas mountains. The traveler of to-day little realizes the dangers of these trips, when over very rough roads we made about 100 miles in a day and a night.

I subsequently made this trip six times, up to 1882, and each time I missed an Indian attack by from one to four days. The day I arrived at Silver City news had come of the massacre of 75 women and children at the San Francisco ranch. The fighting-men of Silver City had gone to rescue others. I was the only passenger from Socorro, N. M., until we picked up the soldiers. The next day we came suddenly on 5 of Victorio's band, but we had picked up several miners, so our stage bristled with guns, and we were let alone.

At or near my stone quarry, 6 miles from Socorro, 9 men were killed by the band that, a day or two after, killed George Daly, Lieutenant White, and all their men. During this raid our miners and women at the mine slept in the tunnel. At the Cosette mine I had 35 men at work. These Indian raids had

* Received Feb. 23, 1909.

killed off the freighters, so for a time the only supplies we had were beef, oatmeal, honey, and a few canned tomatoes.

The Cosette mine was on a trachyte dike, breaking through the limestones. The vein carried lead carbonates and horn-silver, as rich as 1 oz. in 6 of ore, averaging $100 in value. Some $20,000 worth had been taken out in the two months during which I was in New York, and I was sent back immediately to test this mine. A very rich semi-opal and hyalite showed that the deposit had been made by hot water. The vein dipped quite steeply for a short distance, then ran nearly flat, and pitched again. The flat parts carried the rich horn-silver, 90 lb. being our largest mass, of which one-sixth was silver. Immediately after this find we broke into a cave of greater depth than we could measure and very hot. It was not over 18 in. wide, and was lined with ore that ran in a small testing-mill from $35 to $52 per ton. For a mill, I had found a flowing stream, by noticing that the prairie-dogs had dug up wet dirt in this desert. It was only 16 ft. down to water. The mine would have paid for a mill, but the parties sold it, as the drop in the ore discouraged them, though for a working-expense of $13,000, and hauling the ore 5 miles, we had taken out more than $8,000 in the testing.

While working here the Indians had raided the country about Lake Valley and killed some of the men. The McEverts ranch covered the mines then known only as possible prospects. One of my men said that Mrs. McEverts wanted to sell the ranch for $5,000, and that there were good signs of ore like ours. I had $5,000 to invest for parties, but the Indians were too close just then to go there, and I would not take it without seeing it. John A. Miller, sutler at Fort Bayard, took a guard of soldiers and looked at the ranch, making a bargain to purchase it at about the price offered to me, thinking of it only as a hay-ranch. Seeing this very rich ore I had found, he set some men to work on similar outcroppings. I assayed the samples his men found. At first these did not run over $42, but in a few days one sample ran about $10,000, and Mr. Miller made me an offer of a half interest if I would put up a small smelter, to cost $20,000. I took the matter up with Capt. W. H. Stevens, of Detroit, owning the Iron-Silver mines of Leadville, who did not accept the proposition.

Shortly after, Mr. Miller told me that he had sold part for $125,000, and the purchasers made me an offer to come back and put up the works, but those with whom I had an engagement would not release me. Mr. Miller told me he received $350,000 for all his interests in this property. I was idle some months under salary at Detroit waiting orders, until sent to Socorro to open up the Torrence mine. I understood that R. Bunsen, of Leadville, had built a smelter at Lake Valley that was not successful on the class of ore they opened into. George Daly and the other officers I met gave me pieces of what they called "Jackson's baby," a solid mass of horn-silver, found, as I think, in the "Bridal Chamber." I have some pieces now.

So many prominent men were killed by the Indians in 1882 that no one would look at any New Mexican property for years, but some day much gold will be obtained from neglected mines. I opened great bodies of $10 gold-ore, but as the expense was $12 per ton we could do nothing with it. I had found out in 1881 that our ore would extract to 90 per cent., with a 10-per cent. solution of cyanide, but with cyanide at $2 per pound it did not look attractive for a $10 ore.

On the trip early in 1880 I was given a 10-day option for $80,000 on 22 claims at Bisbee, including the Copper Queen. The owners had been driven in by the Indians, and brought with them many burro-loads of beautiful ore. I sent four 4-lb. sacks to a Calumet & Hecla stockholder in Boston, but the over-wise mining-man of that company, to whom the matter was referred, whose experience had been limited to the Lake Superior region, said there were no copper-deposits in Arizona that would go down, so I received no aid. Many years after, when I was looking at the ore-stopes above the sixth level, with ore running about 30 per cent. of copper, I wished every young mining engineer could profit by this lesson, and be careful not to make too wide a generalization from a limited experience. The work I laid out on the belief that the copper would go down in Arizona made my clients a very satisfactory return.

Charles R. Keyes, Des Moines, Iowa (communication to the Secretary*):—It is with pleasure that I acknowledge the value and interest of the historical notes on early mining at

* Received Jan. 10, 1910.

Lake Valley, contributed by Messrs. MacDonald[1] and Courtis. In my own paper there was no attempt to enter into this phase of the subject; partly for reason that the intention was to take up this in detail in another connection, partly because there was not space enough to devote to it in a brief geologic paper, and partly on account of the fact that the strictly mining features are quite distinct from the purely geologic characteristics.

As may be inferred from its perusal, the brief historical paragraph prefacing my paper was in reality geologic in its bearing. The events referred to therein relate to the published information concerning the geologic structure. The personages principally mentioned were noted merely to show how the names of several of our most distinguished scientists came to be closely associated with that of Lake Valley.

Far from my intentions was it to slight the memory of any of that brave band who, with lives in their hands, took so active a part in the early development of the mines. The value of their records is perhaps much greater than that of any published notes. The narratives of Messrs. MacDonald and Courtis are as thrilling as many other experiences related of that time and place. Few of us at the present time can appreciate the dangers of that early day, when the fierce Apache all but swept the white man from that part of the country, when renegades from Texas overran the land, and when the outlaws of the nations took their last stand before advancing civilization. In marked contrast do we to-day reach Lake Valley by means of the "iron horse"; and out of a luxurious Pullman palace-car literally step directly into the famous "Bridal Chamber" of the Grande mine.

We are greatly indebted to Messrs. MacDonald and Courtis, especially since their accounts clear up certain hitherto obscure relationships of the various workers in the camp. At this distant day the spoken records are apt to be dimmed and inaccurate. At the same time, by other authenticated records, features are presented from an entirely different view-point. From all, a tolerably complete picture is obtained of the events during the most strenuous period of the Southwestern country. In the near future I hope to present an account of some of the salient historical features of New Mexican mining, in which Lake Valley bears an important part.

---

[1] *Trans.*, xxxix., 850 (1909).

Effect of Humidity on Mine-Explosions.

Discussion of the paper of Carl Scholz, *Trans.*, xxxix., 328.

HOWARD N. EAVENSON, Gary, W. Va. (communication to the Secretary*):—For some time before the publication of Mr. Scholz's paper, I had been collecting data bearing upon its subject, and I now take pleasure in presenting to the Institute my results in the form of a discussion.

It has long been a matter of common opinion among mining-men that mine-explosions are more frequent during the colder, or winter months, than during the warmer ones, and it is a matter of common knowledge among them that the air-ways and road-ways in most mines are much drier and are more inclined to be dusty during the winter than during the summer. In many mines, places which are wet and sloppy in the summer are dry and sometimes dusty during the winter. These changes, of course, can only be due to atmospheric conditions, and are caused by differences in temperature and humidity. Until recently, however, very little attention has been paid to the part that atmospheric conditions may play in mine-explosions, or, rather, to the effect they may have in rendering mine-conditions peculiarly dangerous, and in spreading and rendering more disastrous the effects of what otherwise probably would have been small explosions.

To show the difference in mine-conditions during summer and winter months, I have tabulated in Table I. the results of 24 observations of hygrometric conditions in several different mines at different times of the year.

Mines Nos. 1, 2, and 6 are located in the Pocahontas coal-region, in McDowell county, W. Va., the first one being a shaft-mine and the others drift. Mines 1A and 2A are drift-mines located in Logan county, W. Va. All of these are mines of fair size, but none of them has been in operation more than six years,

* Received Nov. 16, 1908.

and none of them has the amount of air in circulation that it will ultimately require. The conditions given are not extremes of temperatures, but are fair averages for the seasons of the year in which they were taken. The temperatures and humidities are not averages for the entire day, but are averages of two determinations taken at the same time during the day.

TABLE I.—*Humidity-Tests at Various Mines in Southern West Virginia.*

Mine No.	Date.	Outside Air.			Mine Air.			Aqueous Vapor Removed.	Quantity of Air in Circulation.	Amount of Water Removed from Mine.			
		Temperature.	Saturation.	Aqueous Vapor.	Temperature.	Saturation.	Aqueous Vapor.			Per 24 Hr.		Per Minute.	
		F.°	Per Cent.	Gr. Cu Ft	F.°	Per Cent.	Gr. per Cu. Ft.	Gr. Cu. Ft. Air.	Cu. Ft. per Min	Gal.	Short Tons.	Gal.	Lb.
1....	1/24-08	21.8	44	0.591	53.5	97	4.467	3.876	130,000	12,428	51.8	8.6	72.0
1...	2/17-08	29.5	52	0.985	53.8	85	3.955	2.970	128,000	9,377	39.1	6.5	54.3
1....	2/18-08	40.5	44	1.277	53.5	84	3.868	2.591	128,000	8,181	34.1	5.7	47.4
1....	2/19-08	41.0	78	2.305	53.5	85	3.914	1.609	128,000	5,080	21.2	3.5	29.4
1....	2/20-08	28.3	66	1.186	53.3	82	3.751	2.565	128,000	8,098	33.8	5.6	46.9
2....	2/18-08	22.0	79	1.070	53.0	97	4.390	3.320	144,000	11,792	49.2	8.2	68.3
2....	2/19-08	41.8	70	2.130	52.8	98	4.405	2.275	144,000	8,081	33.7	5.6	46.8
2....	2/20-08	25.7	75	1.201	53.0	97	4.390	3.189	144,000	11,326	47.2	7.9	65.6
2....	2/21-08	27.5	76	1.319	52.3	97	4.286	2.967	144,000	10,558	43.9	7.3	61.0
2....	2/22-08	30.0	47	0.910	52.5	97	4.316	3.406	144,000	12,097	50.2	8.4	70.0
2....	2/24-08	32.8	56	1.215	52.7	98	4.390	3.175	144,000	11,277	47.0	7.8	65.3
2....	2/25-08	37.6	72	1.877	52.5	98	4.360	2.483	144,000	8,820	36.8	6.1	51.1
2....	2/26-08	35.5	83	2.156	52.6	99	4.420	2.264	144,000	8,041	33.5	5.6	46.6
6....	2/21-08	38.0	43	1.137	52.8	94	4.225	3.088	200,000	15,234	63.5	10.6	88.2
6....	2/22-08	24.5	68	1.031	50.5	97	4.024	2.993	200,000	14,765	61.6	10.2	85.5
6....	2/24-08	39.0	35	0.961	52.7	97	4.345	3.384	200,000	16,694	69.6	11.6	96.7
6....	2/25-08	29.3	56	1.051	52.4	97	4.301	3.250	200,000	16,033	66.9	9.1	92.9
2A.	Jan., 08	41.0	92	2.719	59.0	94	5.222	2.503	105,000	6,483	27.0	4.5	37.5
2A.	Feb., 08	37.0	91	2.321	58.0	94	5.048	2.727	106,000	7,130	29.7	4.9	41.3
1A.	Mar., 08	54.0	76	3.561	56.0	93	4.665	1.104	102,000	2,777	11.6	1.9	16.1
								Aqueous Vapor Deposited.		Amount of Water Deposited in Mine.			
								Gr. Cu. Ft. Air.					
6....	8/19-08	83.5	57	6.938	60.0	99	5.687	1.251	150,000	4,629	19.3	3.2	26.8
6....	8/20-08	71.0	80	6.592	60.0	99	5.687	0.905	150,000	3,348	14.0	2.3	19.4
6....	8/22-08	72.0	93	7.912	60.2	99	5.726	2.186	150,000	8,088	33.7	5.6	46.8
4....	8/24-08	80.0	58	6.342	59.2	99	5.437	0.805	150,000	2,978	12.4	2.1	17.2

These figures show that during the winter months water is removed from the mines, by the ventilating-current, at rates of from 21 to 69 tons in 24 hr., and that in the summer months it is deposited in the same mines at rates of from 12 to 33 tons in 24 hr. There are extreme differences shown in the amount of aqueous vapor present in the air of the same mine, No. 6, between summer and winter conditions, of more than 100 tons in 24 hours.

Unless one is accustomed to considering the amounts of moisture present in the atmosphere, at all times, these quantities are startling, but the observations can easily be duplicated at trifling expense by any one interested. Can any one doubt that if coal-dust by itself is explosive, as now seems proved, the removal of the amount of moisture shown, from the mine-air, in mines apt to be dusty, may remove the balance from the point on the safety-valve at which nature has set it, and convert a mine ordinarily safe into one liable to cause a terrific explosion, if anything should happen to raise the temperature to the explosion-point of the dust?

With the exception of some data published by D. E. Llewellyn,[1] I have never seen any definite figures giving the numbers of explosions, or of fatalities caused by them, occurring in each month of the year. Unfortunately, the data given by Mr. Llewellyn are confessedly incomplete, and, furthermore, they refer only to explosions occurring in England and Wales, where the atmospheric conditions are entirely different from those existing in this country.

For the purpose of determining the relative occurrence of explosions, I have compiled in Table II. the explosions caused by gas or dust in which five or more fatalities have occurred, and which have been officially reported in Canada, Mexico, and the United States. No explosions or fatalities caused by fires, wrecks, falls, etc., have been included, as it is inconceivable that atmospheric conditions could have affected such occurrences to any appreciable extent. The basis of Table II. is one published by J. T. Beard.[2] All of the figures given by him have been verified with the records and by correspondence, and his table has been very much amplified to make the present one, which, it is believed, is accurate and nearly complete.

TABLE II.—*Mine-Explosions Supposed to be Caused by Gas or Dust, Causing Five or More Fatalities, Officially Reported in North America.*

(From Oct. 2, 1871, to Mar. 28, 1908.)

Date.	Place.	Number of Fatalities.
1871, October 2.	Otto Red Ash, Pa.,	5
1872, September 27.	Diamond, Pa.,	7
1873, May 13.	Intercolonial Coal Co., N. S.,	55

[1] *Coal* (February, 1908). [2] *Mine Gases and Explosions*, p. 189 (1908).

Date.	Place.	Number of Fatalities.
1877, May 9.	Wadesville, Pa.,	7
1878, January 15.	Potts Colliery, Pa.,	5
1878, May 21.	Sydney, N. S.,	5
1879, May 6.	Audenried, Pa.,	6
1879, June 20.	Mill Creek, Pa.,	5
1880, May 3.	Lykens Valley Slope, Pa.,	5
1881, March 5.	Shaft No. 2, Nanticoke, Pa.,	6
1882, May 24.	Kohinoor Colliery, Pa.,	5
1884, January 24.	Crested Butte, Colo.,	59
1884, March 13.	Pocahontas, Va.,	114
1884, August 20.	Greenback, Pa.,	7
1885, February 18.	Vale Colliery, N. S.,	13
1885, October 21.	Plymouth, Pa.,	6
1886, January 21.	Newburg, W. Va.,	39
1886, August 30.	Fair Lawn Colliery, Pa.,	5
1886, September 13.	Marvine, Pa.,	8
1890, February 1.	Nottingham Colliery, Pa.,	8
1890, March 3.	Shaft No. 3, S. Wilkes-Barre, Pa.,	8
1890, April 2.	Slope No. 4, Nanticoke, Pa.,	5
1890, May 15.	Jersey, No. 8, Ashley, Pa.,	28
1891, January 27.	Mammoth, Pa.,	109
1891, February 21.	Springhill Mines, N. S.,	125
1891, October.	Richardson, Pa.,	7
1891, November 8.	Shaft No. 1, Nanticoke, Pa.,	12
1892, July 23.	York Farm, Pa.,	15
1893, January 10.	Como, Colo.,	24
1893, March 27.	Shaft No. 1, Nanticoke, Pa.,	6
1893, April 5.	Shaft No. 4, Edwardsville, Pa.,	6
1893, September 21.	Plymouth, Pa.,	6
1894, November 20.	Blanche, W. Va.,	8
1895, February 18.	Bear Ridge, Pa.,	5
1895, October 7.	Dorrance, Pa.,	7
1895, December 20.	Dayton, Tenn.,	25
1896, February 18.	New Castle, Colo.,	49
1896, March 23.	Berwindsdale, Pa.,	15
1896, October 29.	S. Wilkes-Barre, Pa.,	6
1897, January 4.	Alderson, Ind. Ter.,	5
1897, March 28.	Jermyn, No. 1, Pa.,	5
1897, September 3.	Sunshine, Colo.,	12
1898, September 23.	Brownsville, Pa.,	8
1899, March 9.	Mahanoy City, Pa.,	8
1899, June 16.	Dominion Colliery No. 4, N. S.,	11
1899, July 24.	Grindstone, Pa.,	5
1899, December 23.	Sumner, Pa.,	20
1900, March 6.	Red Ash, W. Va.,	46
1900, May 1.	Scofield, Utah,	200
1900, November 2.	Berryburg, W. Va.,	15
1900, November 9.	Buck Mt. Colliery, Pa.,	7
1901, February 15.	Union, B. C.,	64
1901, April 29.	Alderson, Ind. Ter.,	6

Date.	Place.	Number of Fatalities.
1901, May 15.	Chatham, W. Va.,	10
1901, May 20.	Richland, Tenn.,	20
1901, June 10.	Port Royal, Pa.,	19
1901, September 16.	Spring Gulch, Colo.,	6
1901, October 25.	Buttonwood, Pa.,	6
1901, October 26.	Diamondsville, Wyo.,	32
1902, January 24.	Lost Creek, Iowa,	20
1902, March 6.	Catsburg, Pa.,	5
1902, March 31.	Dayton, Tenn.,	16
1902, May 19.	Coal Creek (Fraterville), Tenn.,	184
1902, May 22.	Fernie, B. C.,	125
1902, July 10.	Rolling Mill Mine, Johnstown, Pa.,	112
1902, August 7.	Bowen, Colo.,	13
1902, September 16.	Algoma, W. Va.,	17
1902, September 22.	Stafford, W. Va.,	6
1902, November 29.	Luke Fidler, Pa.,	7
1903, April 13.	Carbon, Ind. Ter.,	6
1903, June 30.	Hanna, Wyo.,	169
1903, July 15.	Union, B. C.,	16
1903, November 21.	Ferguson, Pa.,	17
1903, December.	Flat Top, Ala.,	5
1904, January 8.	Crows Nest, B. C.,	7
1904, January 25.	Cheswick (Harwick Mine), Pa.,	178
1904, October 28.	Tercio, Colo.,	19
1904, November 18.	Crows Nest, B. C.,	14
1905, February 20.	Virginia City, Ala.,	111
1905, February 26.	Wilcoe, W. Va.,	7
1905, March 18.	Red Ash, W. Va.,	24
1905, March 22.	Princeton, Ind.,	9
1905, April 3.	Ziegler, Ill.,	53
1905, April 20.	Cabin Creek, W. Va.,	6
1905, April 27.	Dubois, Pa.,	13
1905, April 30.	Wilberton, Ind. Ter.,	13
1905, July 5.	Vivian, W. Va.,	5
1905, October 29.	Hazel Kirk, Pa.,	5
1905, November 4.	Vivian, W. Va.,	7
1905, November 15.	Bentleyville, Pa.,	6
1905, December 1.	Diamondville, Wyo.,	18
1906, January 4.	Coaldale, W. Va.,	22
1906, January 18.	Detroit, W. Va.	18
1906, January 24.	Witteville, Ind. Ter.,	14
1906, February 8.	Parral, W. Va.,	27
1906, February 19.	Maitland, Colo.,	14
1906, February 27.	Piper, Ala.,	12
1906, March 22.	Century, W. Va.,	23
1906, April 22.	Cautro, Colo.,	19
1906, July 19.	Huger, W. Va.,	5
1906, October 3.	Pocahontas, Va.,	36
1906, October 5.	Blossburg, N. M.,	15
1906, October 24.	Johnstown, Pa.,	7

Date.	Place.	Number of Fatalities.
1907, January 23.	Primero, Colo.,	24
1907, January 26.	Penco, W. Va.,	12
1907, January 29.	Fayetteville, W. Va.,	85
1907, February 4.	Elkins, W. Va.,	25
1907, May 1.	Whipple, W. Va.,	16
1907, December.	Naomi, Pa.,	35
1907, December.	Monongah, W. Va.,	358
1907, December.	Darr, Pa.,	239
1907, December 16.	Yolande, Ala.,	16
1908, February 7.	Port Hood, N. S.,	10
1908, February 10.	South Carrollton, Ky.,	9
1908, February 27.	Roseta, Mexico,	83
1908, March 28.	Hanna, Wyo.,	59

An analysis of the data of Table II. is presented in Table III.

TABLE III.—*Number of Explosions in which Five or More Fatalities Occurred, and of Fatalities, Caused by Gas or Dust, in North America.*

(From Oct. 2, 1871, to Mar. 28, 1908.)

Month.	Number of Explosions.	Number of Fatalities.	Average Number of Fatalities per Explosion.	Percentage of	
				Explosions.	Fatalities.
January	15	621	41.4	12.9	16.6
February	15	562	37.5	12.9	15.0
March	14	344	24.6	12.1	9.2
April	9	127	14.1	7.8	3.4
May	13	666	51.2	11.2	17.8
June	4	204	51.0	3.4	5.5
July	6	158	26.3	5.2	4.2
August	3	25	8.3	2.6	0.7
September	8	70	8.8	6.9	1.9
October	12	151	12.6	10.3	4.1
November	9	93	10.3	7.8	2.5
December	8	716	89.5	6.9	19.1
Total	116	3,737	32.2	100.0	100.0

It will be noticed that by far the larger proportion of explosions and of fatalities has occurred in the eastern part of the United States, in the Appalachian region, as was to be expected, considering the relative production and number of mines of this section and of the remainder of the country. An analysis of these explosions separately, including in the Appalachian region, Nova Scotia, Pennsylvania, Virginia, West Virginia, Tennessee, and Alabama, is given in Table IV.

TABLE IV.—*Number of Explosions, in which Five or More Fatalities Occurred, and of Fatalities, Caused by Gas or Dust, in the Appalachian Region of North America.*

(From Oct. 2, 1871, to Mar. 28, 1908, by months.)

Month.	Number of Explosions.	Number of Fatalities.	Average Number of Fatalities per Explosion.	Percentage of	
				Explosions.	Fatalities.
January	8	468	58.5	9.4	18.3
February	10	343	34.3	11.8	13.4
March	12	276	23.0	14.1	10.8
April	4	30	7.5	4.8	1.2
May	11	341	31.0	12.9	13.3
June	3	35	11.7	3.5	1.4
July	5	142	28.4	5.9	5.6
August	2	12	6.0	2.4	0.5
September	6	52	8.7	7.0	2.0
October	9	85	9.4	10.6	3.3
November	8	79	9.9	9.4	3.0
December	7	698	99.7	8.2	27.2
Total	85	2,561	30.1	100.0	100.0

To ascertain the average atmospheric conditions over the area covered by the Appalachian region I obtained from the U. S. Weather Bureau the average monthly temperatures and the average monthly actual humidities for each year from 1896 to 1907, inclusive, for the following places: Lynchburg, Va., Pittsburg, Pa., Elkins, W. Va., Knoxville, Tenn., and Atlanta, Ga.; these stations being taken as covering the region fairly and being the ones for which records could best be obtained, and from these data, Tables V. and VI. were compiled.

TABLE V.—*Average Monthly Weight of Aqueous Vapor in Air, 1896 to 1907, Inclusive.*

Grains per Cubic Foot.

Place.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.
Lynchburg, Va.	1.75	1.61	2.50	2.97	4.75	6.27	7.15	7.06	5.53	3.81	2.50	1.81
Pittsburg, Pa.	1.56	1.40	2.16	2.71	4.10	5.56	6.46	6.04	4.96	3.35	2.31	1.64
Elkins, W. Va.	1.53	1.27	2.14	2.54	3.97	5.83	6.20	5.99	4.76	2.97	2.05	1.64
Knoxville, Tenn.	1.95	1.85	2.86	3.22	4.89	6.46	7.21	7.25	5.81	3.90	2.68	2.03
Atlanta, Ga.	2.27	2.06	3.20	3.62	5.11	6.50	7.42	7.49	5.96	4.06	2.98	2.28
Averages	1.81	1.64	2.57	3.01	4.56	6.02	6.89	6.77	5.40	3.62	2.50	1.88

TABLE VI.—*Average Monthly Temperatures for Years* 1896 *to* 1907, *Inclusive.*

Degrees Fahrenheit.

Place.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.
Lynchburg, Va......	37.0	35.8	47.6	55.1	66.3	72.5	77.2	76.0	69.7	57.8	47.4	38.4
Pittsburg, Pa.........	31.5	28.7	41.9	50.6	63.3	70.1	75.1	73.1	67.5	55.9	43.9	33.4
Elkins, W. Va,........	30.5	27.0	41.2	46.7	59.3	65.8	70.2	69.3	63.2	51.9	39.7	31.2
Knoxville, Tenn...	38.9	38.3	50.9	56.5	68.0	74.4	77.4	76.8	71.4	59.5	48.1	39.4
Atlanta, Ga...........	42.6	42.3	53.7	59.2	70.6	76.4	78.6	78.1	73.3	62.5	52.2	43.3
Averages............	36.1	34.4	47.1	53.6	65.5	71.8	75.7	74.7	69.0	57.5	46.3	37.1

If we assume the average mine-temperature at outlet to be 52° F. and the relative humidity of the return-air to be 80 per cent.—conservative figures—the return-air would contain 3.50 grains of moisture per cubic foot when leaving the mine. In the Appalachian region, therefore, moisture would be extracted from the mines in considerable quantities during the five colder or low-humidity months of November, December, January, February and March, and possibly slightly in April.

Table III. shows that during the five low-humidity months, 52.6 per cent. of the number of serious explosions and 62.4 per cent. of the fatalities caused by them, have occurred, and that the average number of fatalities per explosion occurring in these five months is 38.3, against an average of 25.5 for the remaining seven months and of 32.2 for the year.

Table IV. shows that, in the Appalachian region, during the five low-humidity months, or 41.7 per cent. of the period covered by the table, 52.9 per cent. of the number of serious explosions, and 72.8 per cent. of the fatalities caused by such explosions, have occurred, and that the average number of fatalities per explosion occurring in these five months is 41.4, against an average of 17.4 for the remaining seven months and of 30.1 for the year.

These figures indicate that more serious explosions do occur during the colder or low-humidity months than during the warmer ones, but that the disproportion in numbers is not as great as is usually supposed.

To see if this apparent tendency for explosions caused by gas or dust to occur more frequently during the low-humidity months extended to all such explosions, or only to the more serious ones, the following three tables, covering all of the data available, were compiled:

TABLE VII.—*Number of Explosions and of Fatalities, Supposed to be Caused by Dust or Gas, in Mines of Pennsylvania.*

(From 1871 to 1907, Inclusive, by Months. Revised to 1907, 1/04/09).

Month.	Number of Explosions.	Number of Fatalities.	Average Number of Fatalities per Explosion.	Percentage of	
				Explosions.	Fatalities.
January	46	362	7.9	10.5	22.6
February	23	41	1.8	5.3	2.5
March	43	108	2.5	9.8	6.7
April	30	63	2.1	6.9	3.9
May	39	106	2.7	8.9	6.6
June	38	75	2.0	8.7	4.7
July	28	167	6.0	6.4	10.4
August	33	60	1.8	7.6	3.7
September	36	84	2.3	8.2	5.2
October	51	105	2.1	11.7	6.6
November	29	80	2.8	6.6	5.0
December	41	354	8.6	9.4	22.1
Total	437	1,605	3.67	100.0	100.0

TABLE VIII.—*Number of Explosions and of Fatalities, Supposed to be Caused by Gas or Dust, in Mines of West Virginia.*

(From June 30, 1893, to June 30, 1907, inclusive, by Months.)

Month.	Number of Explosions.	Number of Fatalities.	Average Number of Fatalities per Explosion.	Percentage of	
				Explosions.	Fatalities.
January	3	125	41.7	9.7	33.1
February	5	59	11.8	16.1	15.6
March	4	95	23.8	12.9	25.1
April	....	....	....	....	....
May	3	27	9.0	9.7	7.2
June	2	2	1.0	6.5	0.6
July	3	8	2.7	9.7	2.1
August	1	1	1.0	3.2	0.3
September	3	24	8.0	9.7	6.4
October	1	1	1.0	3.2	0.3
November	5	34	6.8	16.1	9.0
December	1	1	1.0	3.2	0.3
Total	31	377	12.2	100.0	100.0

TABLE IX.—*Number of Explosions and of Fatalities, Supposed to be Caused by Gas or Dust, in Mines of Indian Territory.*

(From 1896 to 1907, inclusive, by Months.)

Month.	Number of Explosions.	Number of Fatalities.	Average Number of Fatalities per Explosion.	Percentage of	
				Explosions.	Fatalities.
January	9	10	1.1	10.0	8.7
February	7	13	1.9	7.8	11.3
March	8	3	0.4	8.9	2.6
April	7	34	4.9	7.8	29.5
May	3	4	1.3	3.3	3.5
June	5	4	0.8	5.6	3.5
July	6	7	1.2	6.6	6.1
August	8	4	0.5	8.9	3.5
September	7	3	0.4	7.8	2.6
October	11	9	0.8	12.2	7.8
November	10	10	1.0	11.1	8.7
December	9	14	1.5	10.0	12.2
Total	90	115	1.3	100.0	100.0

Table X., which follows, has been compiled from Tables VII., VIII., and IX.

TABLE X.—*Comparative Numbers and Fatalities of Explosions Occurring During Five Colder Months and Remainder of Year in Pennsylvania, West Virginia, and Indian Territory.*

	Pennsylvania.		West Virginia.		Indian Territory.	
	Nov.-Mar. Inclusive.	Remainder of Year.	Nov.-Mar. Inclusive.	Remainder of Year.	Nov.-Mar. Inclusive.	Remainder of Year.
Percentage of time	41.7	58.3	41.7	58.3	41.7	58.3
Percentage of explosions	41.6	58.4	58.0	42.0	47.8	52.2
Percentage of fatalities	58.9	41.1	83.3	16.7	43.5	56.5
Av. number fatalities per explosion	5.2	2.8	17.4	4.8	1.2	1.4

From the data shown, the following conclusions can be drawn:

1. That in the history of coal-mining in North America serious explosions of gas or dust have occurred more frequently during the five low-humidity months than during the rest of the year, and that the explosions occurring during those months have been more severe and have occasioned more fatalities than the others.

2. That in all explosions of gas or dust causing fatalities in Pennsylvania, during the past 37 years, no more than the average number occurred during the five low-humidity months, but that the explosions occurring during these months caused more fatalities than those occurring during the remainder of the year. It should be remarked that in one of the most severe explosions of recent years, in a low-humidity month, the testimony of the mine-inspector at the inquest states that the air-shaft was at least partly closed by an accumulation of ice at the bottom. If this is so, the cause of this disaster should not be attributed to gas or dust.

3. That in all explosions of gas or dust causing fatalities in West Virginia, during the past 14 years, the average number of explosions was greater, and the average number of fatalities per explosion very much greater, during the five low-humidity months than during the balance of the year. The data for the year ending June 30, 1908, are not yet available, but the serious explosion in December, at Monongah, will materially increase the figures given.

4. That in all explosions of gas or dust causing fatalities in Indian Territory, during the past 12 years, the average number of explosions was slightly greater, and the average number of fatalities per explosion slightly less, during the five low-humidity months than during the balance of the year.

I have no records of relative humidities and temperatures of any places in the coal-fields of Indian Territory, and it would be interesting to have the tabulated data prepared by Mr. Scholz for a comparison with the data for the Appalachian region.

In conclusion, I record my opinion that, whatever may be the effect of humidity on gas- or dust-explosions, nearly all of them can be prevented by effective ventilation, by removing the dust, and by the use of safety, or non-flaming, explosives. I am not a believer in the theory that too much air causes explosions, and I mean by "effective" ventilation the circulation of volumes of air whose velocity must not be less than 200 ft. per min. through the last break-throughs and at the faces of the working-places. If this is done and the dust is properly sprinkled and removed, there will be very few of the "mysterious" explosions in coal-mines, of which we see so much in the papers.

All accidents of this sort are to be deplored, and their prevention should be the constant endeavor of every mining-man, but the fatalities caused by them are very much less in extent than are those occurring constantly around mines, chiefly by falls of coal and slate and from mine-cars, and about which nothing is heard by the outside world until the annual statistics of mining-operations are published. Constant instruction and careful oversight are both necessary to reduce the number of fatalities from these causes.

A. T. SHURICK, Washoe, Mont. (communication to the Secretary*):—This paper is to be considered more in the light of a sequel to, rather than a discussion of, Mr. Scholz's theory of humidity in connection with mine-explosions.

Among the many papers and discussions brought forth as a result of the recent unprecedented record of mine-disasters, the Scholz theory, with its accurate statistical data and its final practical results, must appeal to both theoretical and practical coal-men as the one step backed with substantial results. And yet this theory has one unknown quantity, upon the practical solution of which will depend its acceptance among the coal-engineers of the country. This question, in the form that each mine-manager has probably considered it, is: "What relative humidity will my mine require to make it explosion-proof?"

While a discussion of the causes of explosions will not be attempted here, the fact that some mines are more explosive than others, regardless of the chemical and physical properties of the coal—and also occasionally irrespective of the gaseous character of the mine—has been generally accepted among the coal-engineers. Why this is so we are waiting for the chemists to tell us, as we have been waiting for the reason that coal cokes.

For this reason few managers, some even with extremely dusty and seemingly dangerous mines, will countenance the apparently foolish expenditure of money for an adequate watering-system, until there is some practical evidence of its necessity, with the usual toll of human lives. If the mine-manager can be shown that a certain relative humidity is required to

* Received Mar. 15, 1909.

make his mine non-explosive it gives him something tangible to work from, and would doubtless be universally accepted and acted upon.

To accomplish this, I propose that coal from every mine, or every group of mines, in the country be tested at the explosion-gallery of the U. S. Geological Survey, Technologic Branch, at Pittsburg. Each coal could be tested with the air at different degrees of saturation, until the maximum relative humidity at which it was found explosive had been determined. The cannon used would be a factor of safety in more or less direct proportion of its area to the area of the drill-holes used in the mines when an equal number of inches of powder are used. In very gaseous districts the maximum percentage of gas on any return air-way should be ascertained and a like amount used in the tests. When the maximum degree of saturation requisite for a particular mine has been determined, the size of the plant necessary is readily determined when the minimum relative humidity of the mine for the year is known. Hygrometer tests can be made daily or oftener, as conditions seem to warrant, and the mine-air kept closely to the specified humidity, thus eliminating the indiscriminate sprinkling as practiced in many mines, with its indeterminate results and bad effects on the roof.

Two important objections are conceded to this theory, which may be found impossible to overcome in practice.

First, the impossibility of reproducing exact mine-conditions in the test-gallery. This objection can only be overcome by making the gallery of greater length.

Second, the vaporizer would be an objectionable method of raising the humidity in slow air-currents, as much of the water would necessarily be deposited locally. Mr. Scholz's experiments with vaporizers in the thin seams of Oklahoma, with their necessarily high-velocity air-currents, does not give conclusive proof of their adaptability to mines with large airways.

The Silver-Mines of Mexico.

Discussion of the paper of Albert F. J. Bordeaux, *Trans.*, xxxix., 357.

A. H. Bromly, Zihuatanejo, Guerrero, Mexico (communication to the Secretary*):—The following criticism of what "is offered as a summary which may be found useful by mining engineers," is not dictated by any captious spirit, but in the interest of historical and scientific accuracy.

The States of Queretaro and Oaxaca have produced sufficient silver to warrant more extended notice.

Referring to Pachuca, State of Hidalgo, the somewhat loose and inexact statement is made that "the ore-deposit is an andesite, intermixed with quartz-veins, rich from the surface." A better description would be,—the ore-deposits consist of quartz-veins in andesite.

It is notable that many of the rich shoots worked in recent days did not come to surface. In consequence, Pachuca is often quoted by the ill-informed as a camp in which values improve with depth, the truth being that the veins carry shoots, or *clavos*, some of which were exposed at surface and some not.

The name of "electric plow," applied to the mechanical stirrer of recent use, is certainly not to be recommended. The stirrer, or agitator, consists essentially of a traversing gang-plow, operated by power through shafting, drums, and wire-rope. The fact that in certain plants the motive-power used is electricity does not make it an electric plow. Such a name suggests a machine of which the motor is an essential and integral part; which is not so in this case. A lathe in a factory driven by water-power is not called a hydraulic lathe.

My experience agrees with Dr. Raymond's note as to *azulaque*, which in some districts is also applied to blue copper carbonate.

The El Oro, Esperanza, and Mexico mines should be allocated to the State of Mexico; the Dos Estrellas being in Michoacan, as stated. The El Oro quartz certainly does not appeal to me

* Received Nov. 10, 1908.

as being "sugary," and Caetani and Burt[1] describe it as "very hard, compact," and "in places shows a distinctly banded or wavy structure." This entirely agrees with my own observations of the ores both of El Oro and Tlalpujahua. It would be interesting if Mr. Bordeaux could say if the proportions of gold and silver given in ores and bullion apply to the Dos Estrellas.

In the following paragraph reference is made to the noted mine-operator as Jean de la Borde, and later (p. 365) as Laborde. Being of French origin, one or the other of these surnames may be right; but he was known in this country as José de la Borda, and as such is referred to in important historical and family documents. Whatever his surname may have been "back East," is there any good reason to amend history by changing his Christian name from Joseph to John?

Can Mr. Bordeaux positively confirm the statement (p. 364) that veins in Temascaltepec, Sultepec, and Zacualpan occur in granite? It certainly is not so in Zacualpan, the only camp of the three with which I am personally acquainted. There the principal formation is Cretaceous shales and slates, and the same is generally considered to be the case with Sultepec. I think Mr. Bordeaux must be in error.

Respecting the unfortunate State of Guerrero, Mr. Bordeaux appears to be as entangled as he considers the formation to be. Having a good knowledge of a considerable area of the State, I cannot indorse his view, as far as my experience goes. But from his remarks later as to complicated formations in Oaxaca, it may perhaps be inferred that he means the topography is entangled. In that sense, any one who knows would heartily agree with him.

In the neighborhood of Taxco, Buenavista, Tonalapa, and elsewhere, tertiary volcanics have burst through the Cretaceous shales and limestones. In the first-mentioned district, these volcanics are identical with the rhyolites of the central portion of the State of Guanajuato. While, therefore, certain areas show heavy rhyolitic or andesitic exposures, the statement that "dikes of andesite protrude everywhere" is far too sweeping.

Respecting Taxco, historical rummaging has yielded disas-

---

[1] *Trans.*, xxxvii., 5 (1907).

trous results. Certainly, for several centuries Taxco has not presented any tin-ores, complex or otherwise. Historical legend is often unreliable, but according to such, a tribute of gold was paid to Moctezuma from Taxco Viego, a small village about 6 miles from the present town. After the conquest of Mexico, in 1521, Hernan Cortez marched to Acapulco on an exploring-expedition, and incidentally called in at Taxco Viego to investigate the gold-brick industry. It is recorded that he found a village of goldsmiths, and also tin tokens passing as currency. Some historians say that sufficient of the latter were collected to use in casting some bronze cannon. However, he discovered the existence of silver-ores, and started mining-operations himself, thus founding, in 1522, the first mining-camp to be worked by Europeans on the North American continent, the new town being the present Taxco.

Whether the first Spaniards found any tin-ore is uncertain, but in spite of nearly 400 years of active mining-life, and much diligent search, Taxco has produced no tin-ore within the period of reliable records. If tin was ever found there it probably came from superficial deposits in the rhyolite, similar to those of Guanajuato, described in my paper.[2]

The veins of Taxco vary in width from a mere stringer to several meters. At the contact of the slates and underlying limestone there is often a contact mineralization, usually of base or refractory ores. Where a vein in passing from the slate encounters the limestone it dies out, but often a *manto*, or pocket, is found at the contact, sometimes 10 m. or more in thickness. Within the limestone the mineralization ceases, and beyond a narrow fringe at the contact, the limestone, which extends over a considerable area of the State, is practically barren. The straight limestone, unless cut by igneous rocks, or in the vicinity of such, is worthless.

If the statement that "sometimes the veins are from 1 to 2 m. wide," be read to mean that the average width is such, it may be accepted. Mr. Bordeaux repeats a much-quoted error, viz., "the most important bonanzas are in the limestone." Various authorities and writers may be quoted as saying that

---

[2] Tin-Mining and Smelting at Santa Barbara, Guanajuato, Mexico, *Trans.*, xxxvi., 227 (1906).

the veins of Taxco occur in Cretaceous limestone, so Mr. Bordeaux may fairly be excused; although, curiously, he omits Guerrero in treating of ore-deposits in limestone.

The bulk of the silver-wealth of Guerrero in the past has come from Taxco and other districts, being obtained from veins in calcareous shales and slates overlying the limestone, or contact-bodies between the two formations.

The production of Taxco district, since 1522, may have been $60,000,000 (U. S.) or more. Some consider much more. But to say that José de la Borda took out that amount would be a gross exaggeration. A later owner of the same properties, Vincente de Anza, made a very complete and interesting report to the Spanish Crown, dated 1793, detailing the records of the various mines.

From this report it appears that Francisco de la Borda started working in a large way with improved European methods in 1700, and at his death his brother José inherited what was, undoubtedly, a good paying concern. About 1747 José struck *bonanza* in the San Ignacio, and in the following eight years took out about 2,000,000 *pesos* (say, 1,800,000 *onzas*), one-fourth of which he used in building the magnificent church at Taxco. The mine went into *borrasca* and was abandoned in 1759, being re-opened later by the Anza family. At about this period la Borda appears to have left Taxco for Tlalpujahua and Zacatecas.

The greater part of Guerrero is not "uninhabited and unknown." Certain mountainous and remote zones are sparsely inhabited, and there are sections of the Sierra Madre and coast ranges practically uninhabited and but little known. In some sections the climate is excellent; in others, the reverse. On the coast fringe and ranges, subject to precipitation from the Pacific, water is plentiful; in the interior it is generally scarce.

Ore-Treatment.—It would be more correct to give the life of the commercial application of the cyanide process to gold-ores as 21 years.

We are informed in the last paragraph that the extraction by *patio* is 95 per cent. of the silver and 84 per cent. of the gold, a statement requiring amplification to avoid serious error. The average extraction by straight *patio* work may be taken at, say, from 85 to 90 per cent. of the silver, and *nil* to 25 per cent. of the gold. Generally, the *patio* is found to lose the gold.

In the modern combination method, in which the *patio* is both preceded and followed by concentration, the combined extraction may reach the figure given by Mr. Bordeaux. Allowance should be made for subsequent losses incurred in shipment and treatment of the concentrates.

It is interesting to learn of the cyanide process that while agitation was necessary to "reduce the silver first," by which presumably is meant before extraction could proceed, it is now mainly necessary for breaking up clay-balls.

The chemistry of silver-extraction, both by *patio* and cyanide, is somewhat complicated, as Mr. Bordeaux seems to find, and would perhaps have been better reserved for detailed treatment in a separate paper.

Albert F. J. Bordeaux, Thonon les Bains, France (communication to the Secretary*):—I accept Mr. Bromly's criticism of my paper, except as follows:

1. I did not say that all veins in the Temascaltepec, Sultepec, and Zacualpan districts are in the granite, but that they are in the crystalline schists and the granite, and in the schists for Zacualpan.

2. The Dos Estrellas quartz appeared to me as sugary, quite different from El Oro quartz. The proportion of gold and silver at Dos Estrellas is 200 g. of gold and 300 g. of silver in bullion; and the extraction is 97 per cent. of the gold, 60 per cent. of the silver, according to my notes.

3. I was in error about the Laborde's bonanzas of Taxco. According to my references, the value is not $60,000,000, but 60,000,000 francs.

4. With regard to the treatment of Mexican ores, it will be necessary for me, or for Mr. Bromly, if he prefers it, to write for the *Transactions* a complete description of the cyanide-treatment for silver-ores.

Generally speaking, my paper was intended to be only a tentative classification of Mexican mines, and I am glad to see that Mr. Bromly does not contradict it as such.

* Received Apr. 10, 1909.

Professional Ethics.

Discussion of the paper of John Hays Hammond, *Trans.*, xxxix., 620.

PROF. HENRY LOUIS, Newcastle-upon-Tyne, Eng. (communication to the Secretary*):—I welcome Mr. Hammond's paper as an attempt to give definiteness to the best modern professional practice. Such a codification of our professional ethics is urgently needed, seeing how widely the views of mining engineers differ on some of the points raised; for example, I have no hesitation in performing a class of work, as to which Mr. Hammond seems to entertain some doubts, and, on the other hand, I strongly condemn a practice which he appears to recommend. I have not the least hesitation in making reports for vendors of mining-properties, and am constantly doing so; the best justification for this practice may be found in the fact that such reports have been repeatedly accepted by buyers as giving a correct representation of the facts of the case, which is all that should be demanded from any report. On the other hand, Mr. Hammond advocates that mining engineers should take payment for their reports in shares of a company to be floated upon the basis of such reports. I hold that this is ethically wrong, as even the most honest of men is liable to be unconsciously biased under these conditions. Indeed, I go much further, and maintain that a mining engineer who reports or advises upon mining-properties should never hold any mining-shares at all, or be himself interested in any mine, and this has been my consistent practice for many years. A man who hold shares in a mine, and is called upon to report upon a neighboring property or a similar mine, can never be wholly unbiased; and even if he could be, he would always lay himself open to the suspicion that he was giving a favorable report so as to enhance the value of the property in which he himself is interested, or an unfavorable report so as to stifle competition. It is quite as bad for the manager of a mine or the consulting manager to be interested, because he will have access

* Received Dec. 26, 1908.

to information before it reaches any of his fellow shareholders, and he will certainly be unable to avoid incurring the suspicion, if nothing worse, of turning such knowledge to account on the stock exchange. Of course, I do not suggest that there is the slightest shade of anything dishonorable in speculating in mining-shares, in dealing in mining-properties, or in selling or buying these on commission, but I do say that none of these operations can fairly be indulged in by any one holding the fiduciary position of a consulting mining engineer, who, in my opinion, should never own, directly or indirectly, a single mining-share or be otherwise interested pecuniarily in any mining-property. It will be seen how widely I differ from Mr. Hammond on this very important point, and such divergence of views shows how urgently such papers as his are needed, and how important it is that they should receive full and free discussion, in order that the general opinion of the profession may be ascertained, and, if possible, an agreement to insure uniformity in professional practice may be arrived at.

The Clinton Iron-Ore Deposits in Stone Valley, Huntingdon County, Pa.

Discussion of the paper of J. J. Rutledge, p. 134.

WILLIAM KELLY, Vulcan, Mich.:—In the northern part of Bedford county, Pa., the county immediately south of Huntingdon, the Clinton measures appear along the eastern slope of Tussey mountain near its base. The only iron-ore bed that has been worked is in the upper part of the formation. Along the mountain this seam has a dip of more than 60°. At the outcrop the ore is soft and rich, but with depth it passes often abruptly into an almost pure limestone similar to that described by Dr. Rutledge. The soft ore extends deeper in the cross-gullies than it does under the higher ground between the depressions. The slates adjoining the rich ore have been weathered, but where in juxtaposition to lean ore and limestone they are hard. The alteration of the ore-bed is evidently the result of the conditions of drainage in quite recent geo-

logical time. The agencies that have increased the iron-content have not affected the thickness of the bed. These observations agree in the main with those described by Dr. Rutledge, and tend to support his conclusions.

The discussion of the origin of the Clinton ore is exceedingly interesting from a theoretical standpoint, but commercially it makes little difference whether the ore has been concentrated from an original deposition or whether it has been enriched from outside sources. Both processes require the circulation of water, and the limits of this circulation are the limits of merchantable ore. In the Birmingham district of Alabama the percolating waters have gone to a much greater depth below the present surface drainage-level than in the part of Pennsylvania described by Dr. Rutledge. But this fact, or the fact that there are "islands" of lean ore surrounded by rich ore, does not strengthen either theory; nor, on the other hand, does either theory give assurance of greater depth of enrichment than the other. The depth of the ore depends on the depth to which meteoric waters have circulated, and not on the manner of its origin. Since in most cases the depth to which circulating waters have gone can be determined only by the enrichment of the ore, for these measures beyond the limits of actual exploration we must accept the old Cornish maxim about the occurrence of ore, "Where it is, there it is."

H. S. Chamberlain, Chattanooga, Tenn.:—In connection with the interesting descriptions of the Clinton iron-ores in other States, presented at this meeting, I would only say that the members of the Institute will have an opportunity to inspect the development of these ores at the mines of the Roane Iron Co. in the State of Tennessee. With regard to the depth to which the ores of this formation have been found to carry iron in commercially valuable proportions, I would say only, without further discussion of the matter, that the Roane Iron Co. has gone down on the dip 1,200 ft., which corresponds to 800 ft. vertical depth below the outcrop, and the ore is fully as rich and valuable at the bottom as it was at the surface when the mine was first opened.

Ozark Lead- and Zinc-Deposits; Their Genesis, Localization, and Migration.

Discussion of the paper of C. R. Keyes, p. 184.

E. R. Buckley, Flat River, Mo. (communication to the Secretary*):—Some statements in the paper of Mr. Keyes relative to the nature and formation of the Ozark lead- and zinc-deposits seem to me erroneous and misleading, and I respectfully present the following criticisms, placing Mr. Keyes's original statements in quotations.

Referring to the Ozark dome: (p. 192.) "It is a region that has been repeatedly upraised and planed off, until in the middle portion the oldest known rocks only are exposed." Does he mean that the St. Francois mountains, which have always been considered as lying on the eastern flank of the Ozark dome, are the middle of the dome? Or does he mean that the Cambrian rocks of the generally-accepted middle are the oldest known rocks? Certainly not the latter, and if the former, his idea of the middle of the dome is not in accord with that of others who are familiar with the region.

(p. 203.) ". . . the ore-deposits are definitely associated with, or localized by, geologic structures of some kind that produce in effect basins in which the underground waters are impounded, or their flow retarded." This statement does not appear to be altogether consistent with the facts, since the ore-bodies of the Joplin district, as the rule, occur in those parts of the Mississippian formation which have a tendency to accelerate the flow of ground-water rather than retard it. I might name the open, brecciated ground near Joplin, Webb City, and Oronogo as especially noteworthy illustrations. The ore-bodies, everywhere, occur in positions where there must have been a free and copious circulation of dilute lead- and zinc-solutions in the presence of reducing constituents.

The inequalities in the distribution of the ore-deposits are

---

* Received Mar. 11, 1909.

accounted for in the following manner: (p. 200.) "The effects of crustal deformation have been unequal. In consequence of this, the accumulation of the ore-materials is also more or less unevenly disposed." Thus by implication he rejects all agencies such as brecciated or "open" ground, the presence of organic material, etc., as important in the segregation of the ores. Neither does he consider the possibility of an irregular distribution of the metals in the various systems of underground circulation.

(p. 196.) . . . "the recognition of ores of the first circulation and of a subsequent second concentration need not enter into consideration." What is meant by "first circulation"? Does he refer to "primary concentration," as the expression is generally used?

(p. 199.) "Areally, the lead- and zinc-deposits of the Missouri-Arkansas region are mainly distributed in a belt of greater or less width which borders the basal margin of the Ozark dome and completely encircles it." In referring to the central Missouri district: (p. 204.) "While many minor ore-bodies still remain in this upper zone, the extensive ore-bodies of the central district must be sought at deeper levels than in other parts of the Ozark region—at, or below, the present deep-lying permanent water-level." This "central district" actually lies near the middle of the dome referred to, and the inference one would draw from this statement is that extensive ore-bodies occur here as well as "in a belt" bordering the basal margin of the dome.

Again referring to this district: (p. 204.) "Whatever ore-bodies once existed in this upper zone have been largely removed. This is shown to some extent by the myriads of caverns throughout the region." Because this is a cavernous region it is not to be assumed that it was once a richly-mineralized region.

Where will one find facts warranting the statement that "all of the known mines [in the central Missouri district] are located in certain straight and narrow belts, which are near and parallel to the axes of shallow synclines"? And that "these synclines pitch radially from the center of the Ozark uplift"? (p. 195.) Keyes gives none, and since most of the recent work in central Missouri has been done by the Missouri Bureau of Geology and

Mines without developing any such relation, his generalization is open to serious doubt.

(p. 230.) "Thus, as the Coal Measures margin retreated down the slope the ore-belt also continually migrated in the same direction, until its present position was reached." This statement hardly appears consistent with those relative to the occurrence of "extensive ore-bodies" "at deeper levels" in the central district.

Keyes lays very great stress upon the occurrence of a broad syncline trending westward through the so-called Joplin district, combined with SE–NW. synclines, of which he says there are no less than four, conforming to major ore-runs near Joplin, Carthage, Galena, and Cartersville. He simply makes the statement that these synclines exist. There is no proof of it in his paper, and I know of none elsewhere available.

He speaks of the intimate relations existing between the Coal Measures remnants and the ore-deposits of the southwestern Missouri district as "accidental associations." (p. 216.) If this is true, the "accidental associations" are so numerous as to justify some hesitancy in accepting the statement.

(p. 230.) "The genetic relationship of ore-runs to buried relief-features at the base of the Coal Measures does not obtain." This statement is not accompanied by proof, and its truth should be demonstrated before being accepted.

Although one would scarcely infer as much from Keyes's paper, the demonstration that Bain lacked proof of the existence of major faulting in the Joplin district was first published in the Geology of the Granby Area.[1] Keyes evidently concludes, as others have done, that because faulting is not associated with the ore-deposits of the Joplin district, it is safe to conclude that this structure has no relation to the ore-deposits of other districts in this region. As a matter of fact, the ore-deposits of southeastern Missouri are closely associated with faults, which have evidently played an important part in localizing the deposits of galena. The detailed facts upon which this statement is made are given in a work now in press.[2]

[1] *Report of the Missouri Bureau of Geology and Mines*, Second Series, vol. iv. (1905).
[2] *Idem*, vol. ix. (1909).

Keyes refers to the ore-bodies of the southeastern Missouri lead-district: (p. 195.) "In the southeast Missouri district the unconformity-plane marking the base of the Cambrian terranes seems to have largely controlled the localization of the great bodies of disseminated ores." . . . (p. 211.) "Many of the principal ore-bodies now lie near the bottoms of these ancient drainage-troughs. It seems probable that most, if not all, of the ore-deposits of the district will be eventually found to have some direct connection with the courses of the old troughs. Observations bearing upon this very point were published nearly 15 years ago, and since that time practical tests have fully confirmed the original working hypothesis." . . . (p. 225.) "In the southeastern Missouri lead-district the localization of the ore-bodies appears to be chiefly influenced by the character of the pre-Cambrian channel-ways corraded out of the still older granites." . . . (p. 229.) "In the southeastern district the general conditions are also like they are in the southwestern area. Corresponding to the warped surface is the uneven erosion-plane of unconformity at the base of the Cambrian rocks. Instead of porous zones caused by brecciated cherts and limestones are the porous dolomites."

I do not enter into a discussion of the ore-deposits of the southeastern Missouri district, since I have already covered this subject in detail.[8] If Keyes were in any degree familiar with developments during the last 10 years, he would hardly make the statements above quoted. Winslow pointed out long ago that the ore-bodies of the Bonne Terre and Flat River areas lie in a pitching trough, but my investigations have not disclosed any relation between the minor troughs and the localization of the ore-bodies. The position of the ore-bodies in this pitching trough depends upon several other factors, including faulting, jointing, and sedimentation.

In recapitulating: (p. 230.) "The geographic distribution of the main ore-deposits is circumscribed, the belt in which they are confined forming a continuous circle around the base of the great dome." This has yet to be proven. Thus far there have actually been found only two "main ore-deposit" districts, one in the southwestern part, including Jasper, Newton, Lawrence,

[8] *Report of the Missouri Bureau of Geology and Mines*, Second Series, vol. ix. (1909).

and Greene counties, and the other in the southeastern part of the State, including St. Francois, Washington, Franklin, and Madison counties. The remainder of this "continuous belt" is dotted with prospects, the combined output from which will not equal that of a single mine in St. Francois county. Of the so-called "continuous belt," 75 per cent. is territory which has not yet been proved to have noteworthy deposits of either lead or zinc. Such deposits as occur over this portion of the belt are of little, if any, greater importance than hundreds of others scattered irregularly over the entire Ozark region of Missouri.

(p. 230.) "Primary source of the ore-materials is a factor the importance of which has been very greatly overestimated, and is of no significance in practical mining-operations." There are many pre-Cambrian erosion-basins in southeastern Missouri similar in all respects to that in which occur the disseminated-lead deposits of St. Francois county. They are in all respects, except in the matter of faulting and source of ground-water, under the same conditions, but diamond-drilling seems to indicate that they do not contain deposits of lead. How would Keyes account for this condition?. If he were operating in this district I am sure he would regard faulting, character of sedimentation, and primary source of the ore-materials as of primary importance.

(p. 221.) "The diffused metallic content of the Ozark rocks cannot, therefore, be regarded as derived in any way from the old crystalline basal complex now exposed in the region." If it did not come from the removed portion of this pre-Cambrian complex, where did it come from? From distant areas? Perhaps, in part; but it must be remembered that the rocks of the Palæozoic succession in Missouri are essentially near-shore deposits, and it must be conceded that distant continental areas contributed only subordinate amounts of material to their mass.

Keyes uses a remarkable series of illustrations. To my knowledge they do not represent a single concrete condition in the lead- and zinc-districts of Missouri. Where is the "syncline on the Osage river," Fig. 3; "the unconformity-trough at Doe Run," Fig. 5; the "warp-sags" of Fig. 6; the "silted-up cavern," near Aurora, Fig. 7? Where in the disseminated-lead district do the ores occur as shown in Fig. 10? What is the basis upon which the author worked out the conditions

shown in Fig. 18? Fig. 8 is referred to as "a very detailed picture of the geologic structure in a direction parallel to the margin of the Ozark dome and transverse to the distinct minor flexing." If this is a detailed section, I should like to see a generalized section. Some of the men who have been studying the ore-deposits of the Ozark region for a number of years would be glad to know the exact localities from which these drawings were made. Take Fig. 10, for example; any one who has ever been in the disseminated-lead district knows that the ore now being mined occurs above the Lamotte sandstone, and that only where the sandstone is absent does it rest upon the granite. Yet Keyes represents it as occurring at the base of the Lamotte sandstone.

The man engaged in mining-pursuits is prone to criticise the geologist for publishing theoretical dissertations which have little foundation in fact. Mr. Keyes's paper contains practically no facts relative to the lead- and zinc-deposits of the Ozark region, and without them the theoretical discussion is only of value in so far as the reader may believe the unqualified statements. It is easy enough to deny the existence of certain conditions, stratigraphic and structural, but it is another matter to prove this denial. Likewise, it is easy to assert the existence of certain phenomena, but an entirely different proposition to prove their existence.

Vanadium-Deposits in Peru.

Discussion of the paper of D. Foster Hewett, p. 274.

JAMES F. KEMP, New York, N. Y.:—Mr. Hewett's paper is one of exceptional interest, because it not only adds an important contribution regarding one of the rarer, valuable elements, but also because it introduces compounds hitherto unknown and in associations no less novel. While we have occasionally discovered and recorded asphaltite in metalliferous deposits, for example in the Joplin and Granby districts, we have not seen a metallic ore in a deposit essentially asphaltite. The source and method of introduction afford a subject well worth serious reflection.

Vanadium was first discovered and recognized as a distinct element in the slags obtained in smelting the titaniferous magnetites at Taberg, Sweden. For a long time, therefore, we looked upon the titaniferous magnetites as its home. Analyses proved, years ago, that it was present in the ores of this type in New Jersey and in the Adirondacks. It was quite rarely determined, but when looked for it was, I think, invariably found, although the amount seldom exceeded 0.5 per cent. of V_2O_5. In just what combination it exists in the titaniferous ores is uncertain. One might imagine V_2O_3 replacing some of the Fe_2O_3 of magnetite; or, since in the lead series vanadates and phosphates are closely akin, one might wonder if a lime-vanadate could take the place of apatite. Yet no lime-vanadate has been found in nature, and apatite itself is usually rare in the titaniferous ores. When percentage-curves are plotted for a series of analyses of the ores, the line of vanadic oxide shows a curious sympathetic behavior with the line of chromic oxide;[1] but the data are somewhat limited, and no compound has suggested itself which throws light on the matter.

Dr. W. F. Hillebrand has discussed the occurrence and distribution of vanadium in nature, and concludes that it favors the moderately basic eruptives.[2] In these it may possibly replace the Fe_2O_3 of the silicates, but it does not favor richly magnesian rocks, presumably because there are few sesqui-bases in them to replace. H. S. Washington has also given a brief summary of its distribution in nature, and reaches the same conclusion.[3]

For many years vanadium has been known as a constituent of the so-called coals of Peru and Argentina. In Hillebrand's paper, a contribution by A. A. Hayes is cited in which Dr. Hayes, writing in 1875, quotes Thorpe's *Dictionary of Chemistry*. The last named, obviously of still earlier date, states that a coal from Peru contained 0.45 per cent. of V_2O_5, and that two samples of ash gave 38.5 and 38 per cent. As a constituent of Argentine coals, it is mentioned in the *U. S. Consular Report*

[1] *Nineteenth Annual Report, U. S. Geological Survey*, Part III., Pl. LV., opp. p. 394 (1897–98).

[2] W. F. Hillebrand, Distribution and Quantitative Occurrence of Vanadium and Molybdenum in Rocks of the United States, *American Journal of Science*, Fourth Series, vol. vi., No. 33, p. 209 (Sept., 1898).

[3] *Trans.*, xxxix., 756 (1909).

for 1894, p. 176, and three or four years later I received a letter from R. S. McCaffery, at the time chemist at the smelter in Casapalca, Peru, announcing that he had found notable percentages in the ash of a coal used at the smelter. It was apparently supposed at this time that the so-called coals were of the usual sedimentary types, and it was always a puzzle as to the source of the vanadium. Mr. Hewett has now shown that they are all asphaltites.

Asphaltite-veins are generally believed to have been formed by the entrance of a heavy petroleum, with an asphalt base, into a fissure. Naturally, we would ascribe to the petroleum a source in some oil-pool below. This explanation would suggest itself for the case described by Mr. Hewett. With all the reservation proper to one who has not seen the district, and with full appreciation of the careful work done by Mr. Hewett, who rather favors a derivation of the vanadium by segregation from the neighboring sedimentaries, the uprising petroleum bringing with it the patronite appeals to me rather strongly. This method of vein-filling was trusted for the grahamite-vein of West Virginia, and an oil-pool was subsequently found at a depth of 1,500 to 1,600 ft. by drilling near it.[4] It would be of much interest to know if vanadium sulphide is soluble in these heavy, asphaltic oils, and especially in the sulphurous varieties. Oils must traverse extended sections of rock before gathering in quantity and may thus pick up vanadium. We may note that the moderately basic rocks, dolerite and diabase, the former in a laccolith, the latter in a dike, are both reported near the asphaltite-vein.

Asphaltite is not unknown in deposits of the metallic ores, as has been remarked above. It may be that when graphite appears, as it does in the great sulphide-veins at Ducktown, Tenn., it represents some original hydrocarbon. But a heavy oil as a solvent of a metallic or semi-metallic sulphide, and as the vehicle of its introduction, is something new.

[4] I. C. White, *Bulletin of the Geological Society of America*, vol. x., p. 278 (1898).

Pan-Amalgamation: an Instructive Laboratory-Experiment.

Discussion of the paper of Messrs. Hofman and Hayward, p. 382.

E. A. H. TAYS, San Blas, Sinaloa, Mex. (communication to the Secretary*):—The results obtained by Messrs. Hofman and Hayward in their experiments, proving that a low percentage of copper sulphate with a variable percentage of salt, depending on the ore, gives the best results, confirm practical mill-work which I did in 1895. I have none of my notes, taken at the time, to refer to, so have to rely solely on memory, which precludes the conciseness that is alway desirable.

In 1894 I took charge of the plant of the San Rafael Company, near Chinipas, Chihuahua, Mex., which at that time was treating a very refractory ore carrying about $15 in gold and from 20 to 25 oz. of silver per ton, the bullion being about 300 fine in gold. The silver in the ore was in the form of a sulphide.

Although carrying an average of 0.75 oz. of gold per ton, free gold could rarely be seen by panning, but upon roasting a sample of the ore the gold became visible at once—a result which was discovered by chance and used to advantage later.

In order to utilize the old mill to the best advantage, it was decided to use pan-amalgamation after roasting. The roasting, done in a lime-kiln, was similar to the ordinary "burning" operation. The roasted ore was trammed to the mill-bins and crushed by 10 stamps to pass a 30-mesh screen. The pulp was de-watered and fed to four 1-ton pans.

My recollection is that upon starting the mill-work, the charge contained 1.5 lb. of copper sulphate and 5 per cent. of salt per ton.

From the start the extraction was unsatisfactory and the loss of quicksilver was very large. Experiments were then carried out on a working scale. First, the quantity of copper sulphate was increased 4 oz. at a time; then it was reduced

* Received July 12, 1909.

at the same rate and notes carefully recorded. To my surprise, when the charge of copper sulphate was lowered from the original 1.5 lb. per ton, the percentage of extraction increased, and at 4 oz. per ton of charge the best extraction was made. The quantity of salt was then varied, and 5 per cent. was found to give the best results with the 4-oz. charge of copper sulphate.

When I explained what I was doing to the metallurgist of a nearby plant, he asked, "Why do you use any copper sulphate at all?"

The nature of the ore may be judged by the fact that with only 4 oz. of copper sulphate per ton of ore treated the loss of "quick" was still 1 lb. per ton of ore treated.

George W. Riter, Salt Lake City, Utah (communication to the Secretary*).—In their paper, Messrs. Hofman and Hayward deal with a branch of silver metallurgy that is on very uncertain ground, both as a commercial process and as a metallurgical science; moreover, its field is limited to few localities and to peculiar conditions. The literature on the subject is incomplete and full of conflicts. The paper under discussion clears up no disputed points; on the contrary, because of hasty generalization, its tendency is towards further confusion.

In Table I., the authors assume that each test-lot of 1,800 g. of ore contained exactly 5.499 g. of silver, and that any variation in the quantity of silver recovered, however slight, should be ascribed to variation in the time of grinding. It is barely possible that the ore was mixed so uniformly as to justify this assumption; but those of us who know how difficult it is to get uniform samples of any ore in which silver mineral is present in rich particles will have a doubt on this point.

The unlikelihood of getting absolutely uniform test-lots of ore also reflects on the data in Table II., on the effect of varying the amount of salt in a pan-charge. The authors say: "The salt series was the last one that was investigated." This again raises a question as to whether the ore was split into test-lots at the time of sampling, or whether it was kept in bulk in such a way as to permit mechanical concentration of values before the end of the experiments.

* Received Feb. 3, 1910.

The conclusion following Table II.: "There is no reason, therefore, for going beyond 6 per cent. of salt," is hardly to the point. If we assume that salt is of benefit only when in solution, it follows that the relation between the quantity of salt and the quantity of water is of more consequence than the relation between the quantity of salt and the quantity of ore. Water will hold 26 per cent. of salt at the point of saturation; then why add 180 g. of salt to only 500 cc. of water, as was done in the experiments detailed by the authors?

In discussing the results from the use of blue vitriol, the authors overlook, as so many writers have done, the chemical effect of metallic iron, an excess of which is always presented by the pan-parts and liners. It can be demonstrated that the blue vitriol ceases to remain in solution after coming in contact with the pan-iron, the copper being completely precipitated in metallic form, and being afterwards taken up by the mercury.

In every pan-charge to which salt and blue vitriol have been added, soluble chlorides and sulphates of iron and of sodium will be found. These secondary compounds do act in some beneficial way in the treatment of sulphides of silver, and a quantitative study of their effects might open up a fruitful field. The statement: "The addition of blue vitriol to the pan, as shown in Table V. and Fig. 9, has no beneficial effect whatever; on the contrary, the extraction decreases," is apt to be misleading to the casual reader who fails to note that in the ores used for experiment practically all of the silver was in free state.

Several years ago, while in charge of a large pan-mill, I made a long series of test-runs in a 30-in. experimental pan as well as in the regular 5-ft. mill pans, sometimes substituting cement and wood for the iron pan-parts, and using nearly every common mixture of ores and reagents. Careful sampling of each test-lot of ore, both before and after treatment, was an essential feature of the tests; and, sometimes, the samples were screened into sizes before assaying. Moreover, by taking samples from the amalgamating-pans at varying stages of the process, and then subjecting the filtrates from these samples to chemical analysis; also, by analyzing the particles of amalgam obtained from these periodic samples, some additional light was obtained. Many of the analyses were only qualitative, and on account of

the closing-down of the mill for lack of ore, the investigations were interrupted before the series of tests was complete. Unfortunately, the notes of this work are not within reach. A few things, however, seemed clear:

1. Iron and mercury are the most important reagents in the pan process.

2. In some cases, the mercury is quickened by the addition of a small quantity of freshly precipitated metallic copper. This is one of the things ultimately resulting from the use of blue vitriol.

3. Salt (sodium chloride) solution serves as a solvent and wash, rather than as a reagent, and is most valuable when hot and concentrated. It helps to keep the mercury clean. As a solvent for silver chloride, it gives metallic reducing-agents a better chance to act on mineral in that form.

4. Soluble iron salts, and other secondary products that come from the use of salt and blue vitriol, are of considerable value in the treatment of ores containing silver sulphides.

5. Free acids sometimes assist in the treatment of ores containing native silver sulphides, but are of no benefit with native silver chlorides, and are positively detrimental when the ore contains any lead carbonate or other oxidized lead mineral.

It is not safe to lay down any hard and fast rules, because ore from every mine has its own peculiarities, and numerous unknown factors are introduced locally, the results of which can be determined only by careful analysis, followed by painstaking working-tests. The student of this branch of silver metallurgy, who wishes to save himself much wearisome labor without being led into wrong paths, would do well to consult the works of Percy[1] and Collins.[2] Dr. Percy has sometimes been called the father of English metallurgical literature. His work in the field of silver metallurgy, although only a chapter in his career, commands our admiration because of the exhaustive attempts to get at fundamental principles. His own experiences, as well as the work of former experimenters, are set down in a way that all may imitate with profit and few

[1] *Metallurgy: The Art of Extracting Metals from Their Ores. Silver and Gold.—Part I.* By John Percy. (London, 1880.)

[2] *The Metallurgy of Lead and Silver: Part II.—Silver.* By Henry F. Collins. (London, 1900.)

will succeed in equaling. In beginning one of his prefaces, he says: "Of all the branches of metallurgy, that of which silver forms the subject is, in my opinion, the most extensive, the most varied, and the most complicated."

H. O. HOFMAN and C. R. HAYWARD, Boston, Mass. (communication to the Secretary*):—The discussion of our paper by George W. Riter is of considerable interest in that it shows how omissions on the part of a writer render obscure certain parts of a paper, and how unsatisfied expectations on the part of the reader lead him to put a wrong interpretation upon what has been written.

Taking up the criticisms in the order in which they have been made:

The paper is accused not only of failing to clear up disputed points in pan-amalgamation, but of committing the fault of increasing the confusion that may still exist, in explaining the reactions that govern the process. The main object of the paper was to show that pan-amalgamation can be made to serve as a valuable typical experiment for teaching a student how to adapt a metallurgical process to the treatment of a given ore. At the same time this laboratory-experiment supplements the class-room exercise; in addition, it interests the student, in that the results obtained are quantitative and of such a character that they can be used as a basis for large-scale treatment of the ore under consideration; and lastly, as carried out at the Massachusetts Institute of Technology, the summarizing of the results of a series of ten experiments is accomplished in a few days, before the attention of the student is diverted by other metallurgical work. The study of disputed points in pan-amalgamation did not come under consideration at all. In fact, if this had been the object, the manner of going to work would have taken an entirely different character. An example of students' work along this line is given later.

The supposition in the paper that each test-lot of 1,800 g. of ore contained exactly 5.499 g. of silver is attacked, and the suggestion is made that this was improbable. As the method of preparing the ore for the tests was not given, this criticism is justified; it will disappear, however, when the manipulations are

* Received Mar. 26, 1910.

described. The lot of ore was crushed to pass a 30-mesh sieve, mixed thoroughly on the crusher floor, and sampled by fractional selection, every third shovel being reserved, until the lot was reduced to about 50 lb.; this was further cut down by means of a split-shovel, etc., to furnish two 5-lb. samples. These were assayed by crucible- and scorification-methods, and the results were corrected by determining the losses due to scorification and cupel-absorption. The ore was packed into air-tight wooden boxes, each holding about 60 lb., and stored. Whenever a box was taken out for class-work, it was emptied and the contents were thoroughly mixed. The procedure in handling the ore justifies the assumption that any uneven distribution of silver due to a possible unmixing in storage was corrected before the ore was charged into the pan, and the acceptance of the original assay, and with it the presence of 5.499 g. of silver in 1,800 g. of ore, was therefore warranted.

The third objection, that an excess of salt was used over that which the water could dissolve, holds good, and we plead guilty.

In the paper the statement was made (p. 526) that it did not seem clear why the extraction in silver, high with an addition of 6 per cent. of salt, should decrease with 10 per cent. and then rise again, and that the anomaly required further investigation. In a recent series of tests, the highest percentage of salt was given to pan No. 1 and no salt to pan No. 10, instead of having the reverse order as usual, because it was suspected that the fall in extraction might have something to do with the working of the pans. We were led to this idea, because with the style of pan formerly used in the laboratory, some pans always gave better results than others, but in the present pans no such discrepancies had been noticed, as the pulp-current always appeared to be uniform. In Fig. 1 the dotted curve represents the line given in the original paper; the full-drawn curve shows the results obtained by the last series. Here, as was expected, the extraction increased quickly with an addition of from 1 to 6 per cent. of salt, then more slowly with from 6 to 10 per cent., and remained practically unchanged when more than 10 per cent. was used. The quicker rise of the new curve and its general position above the older one are probably due to the time of grinding having been changed from 1.5 hr. to 1 hour.

As to the use of blue vitriol in an iron pan, the fact that iron

precipitates copper was not overlooked; it could not well be, considering the weight and copper-content of the retort-silver obtained by these tests. Nevertheless, experiments were carried out with an addition of blue vitriol, as, according to practically all reliable literature on pan-amalgamation, this salt is one of the first reagents with which to experiment for an increase in the yield of silver. The conclusion drawn from the results, that blue vitriol had no beneficial effect, in fact decreased the extraction with the ore under consideration, is not misleading, but absolutely exact.

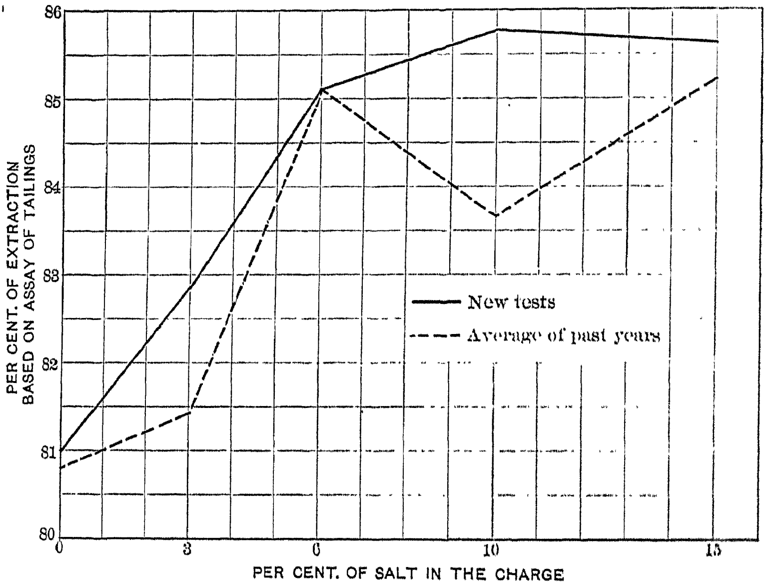


FIG. 1.—CURVES SHOWING EFFECT OF VARYING THE PERCENTAGE OF SALT.

We may add that, in connection with the tests with blue vitriol in iron pans, experiments were carried on in copper pans of the same construction, the copper containing 1 per cent. of silicon in order to reduce the wear. As grinding in copper pans is not permissible, the pulp was previously ground to pass a 100-mesh sieve. The extraction of silver based on the tailings-assay was unsatisfactory; basing a yield on the silver recovered in the amalgam was not feasible on account of the practical impossibility of cleaning-up the pan. Work in copper pans was therefore dropped. If pan-amalgamation were a live issue,

it might be worth while to try the bronze pans as used in the Franke *tina* process at the Huanchaca mine, Bolivia.[1]

The results of Mr. Riter with the ore that he was treating in a large pan-mill are of interest as far as they go; they would have been more valuable if he had told us more about the character of the ore. However, he falls into the error of making "hasty generalizations," which tend "towards further confusion" of a subject that "is incomplete and full of conflicts"—just the points that we avoided, and that he read into our text, although they are not there. In carrying on a metallurgical investigation, accurate conclusions can only be arrived at if a test or a series of tests is so planned as to determine the status of not more than one variable. As soon as you ask your experiment to give an answer at the same time to more than one question, you are sure to go astray.

We can heartily indorse the recommendation of the works of Percy and Collins on the metallurgy of silver. All the books of Percy are of permanent value, because he studied critically the metallurgical chemistry of the metals he was discussing and supplemented by original experiment any gaps that he found in his study. Even if Percy's *Silver and Gold* was somewhat out of date as regards practice when it appeared in 1880, in our day we go back to it when a question as to the metallurgical behavior comes up, to see whether he has said anything about it. Of Collins's *Metallurgy of Silver* we are awaiting a new edition, in which we may expect to find the new available material carefully chronicled and sifted, as is the case in the original work.

As the conclusions of Mr. Riter with regard to the treatment of a certain ore in an iron pan mention the behavior of silver sulphide, some data taken from the records for 1899 of the metallurgical laboratory of the Massachusetts Institute of Technology may be given to supplement the general knowledge available at present. The leading statements regarding the behavior of silver sulphide are those of Percy,[2] Schnabel,[3] Adams,[4]

---

[1] *Engineering and Mining Journal*, vol. xxxviii., No. 8, p. 121 (Aug. 23, 1884).

[2] *Metallurgy: The Art of Extracting Metals from Their Ores. Silver and Gold.*—Part I., p. 74 (London, 1880).

[3] (Translated by H. Louis) *Handbook of Metallurgy*, vol. i., p. 743 (1905).

[4] *Engineering and Mining Journal*, vol. xi., No. 15, p. 233 (Apr. 11, 1871).

Huntington,[5] and Rammelsberg.[6] Percy says that silver sulphide subjected to the action of an aqueous solution of sodium chloride with access of air gives no silver chloride. Schnabel records that silver sulphide is slowly decomposed by quicksilver, with the formation of mercuric sulphide, and that the decomposition is more rapid in the presence of iron, especially upon heating. J. M. Adams found that an addition of salt to the pan did not chloridize the silver in the ore, but had a stimulating effect upon the extraction, as the yield in silver was always higher when salt was present than when it was absent. Huntington's work shows that quicksilver extracts from silver sulphide in the presence of sodium chloride, sand, and water about 87 per cent. of the silver, while the yield is only 29 per cent. in the absence of salt. Rammelsberg found that silver sulphide was decomposed only slowly by quicksilver in the presence of water at 100° C.; the yield of 12 per cent. was increased rapidly to 95.2 per cent. with the addition of iron.

In the experiments carried out in 1899, the finely-divided silver sulphide was prepared[7] by fusing together silver, sulphur, and carbonate of potash and leaching the cake in the water. It may be added that the sulphide obtained is a slimy mass, which settles very slowly and is extremely difficult to wash completely. Some silver sulphide was prepared by precipitation from a nitrate solution by means of hydrogen sulphide. No difference was noticed in the behavior between the sulphides prepared in the dry and in the wet way. An artificial ore was made up by mixing 60 per cent. of quartz, ground through a 40-mesh sieve, and 40 per cent. of fire-clay. This proportion was chosen, as it proved to furnish the most satisfactory pulp-current. Enough silver sulphide was added to the mixture to furnish an ore assaying 100 oz. of silver to the ton.

The pan-charges were made up of 1,200 g. of quartz, 800 g. of clay, 800 g. of water, and 400 g. of mercury. The time of amagalmating was 90 min., and the temperature of the pulp about 70° C.

1. Amalgamation in a copper pan with hydrant-water gave

---

[5] *Engineering and Mining Journal*, vol. xxxiv., No. 12, p. 150 (Sept. 16, 1882).

[6] *Zeitschrift für das Berg-, Hütten- und Salinen-Wesen im Preussischen Staate*, vol. xxix., p. 191 (1881); *Engineering and Mining Journal*, vol. xxxii., No. 22, p. 354 (Nov. 26, 1881).

[7] Berthier, *Traité des Essais*, vol. ii., p. 781 (1834).

65 per cent. extraction; with hydrant-water and 1 per cent. by volume of H_2SO_4, sp. gr. 1.84, gave 68 per cent. extraction; with hydrant-water, and salt to the amount of 5, 10, and 15 per cent. of the weight of the ore, gave, respectively, 84, 94, and 96 per cent. extraction.

2. Amalgamation in an iron pan with hydrant-water, and salt to the amount of 5, 10, and 15 per cent. of the weight of the ore, gave, respectively, 37, 70, and 63 per cent. extraction.

The results show that acidifying the pulp increases slightly the yield in silver, that an addition of salt has a very favorable effect, and that copper acts more energetically than iron.

3. Amalgamation in a copper pan with distilled water—the quartz and clay having been first digested with hot distilled water until all soluble salts had been extracted—was carried on to see whether the decomposition of the silver sulphide was due to chemical or to galvanic action. The electric conductivity of the water after a charge of water, mercury, and quartz-clay mixture had been worked was found to be 28 times as great as the conductivity of the original distilled water, showing that the preliminary digestion had not been sufficient, and that in the amalgamation some salts had gone into solution; an addition of silver sulphide to the pulp and working again under standard conditions, increased the conductivity only three times above the last figure, proving that only a small amount of the sulphide had been rendered soluble. The conclusion is that the decomposition of silver sulphide is mainly chemical and only slightly electrolytic.

4. Amalgamation in a copper pan with hydrant-water, salt to the amount of 10 per cent. of the weight of the ore, and an addition of 5 and 10 per cent. of galena, gave, respectively, 88 and 85 per cent. extraction; 5 and 10 per cent. of chalcopyrite gave 80 and 85 per cent. extraction, showing that the presence of these base-metal sulphides has a harmful effect.

The data furnished are not sufficiently complete to settle definitely the manner and the rate of decomposition of silver sulphide, but they show the trend of the reaction and indicate the yield that may be expected under conditions resembling those of the experiments. The subject has not been carried further, as the interest in pan-amalgamation has decreased in recent years. The manner of carrying on experimental work of this kind retains its original value.

Pressure-Fans vs. Exhaust-Fans.

Discussion of the paper of Audley H. Stow, p. 398.

R. V. NORRIS, Wilkes-Barre, Pa. (communication to the Secretary*):—Mr. Stow's paper presents a series of arguments, numbered from 1 to 18, concerning the relative merits of four systems of colliery-ventilation, designated by the author as "plain pressure-ventilation," "plain exhaust-ventilation," "obstructed pressure-ventilation," and "obstructed exhaust-ventilation." These arguments are subsequently compared by means of tables, in which an arbitrary numerical value is assigned to each, and which consequently serve no other purpose than that of showing his opinions and predilections, as indicated by his assignment of the said numerical values. In addition to this general summary, Mr. Stow intimates his preference for a fifth method of colliery-ventilation—namely, what he calls "alternating pressure-ventilation," which consists in the running of the fan at "pressure" during working-hours, and as an "exhaust" between shifts—a system which he proposes as, perhaps, new, yet which he thinks deserves recognition "as a standard form."

This proposed "standard form" is not to be found in practice; and I think it would be impracticable in gaseous mines of large extent, for the following reasons:

1. It takes considerable time to reverse the air-current in a large mine; and the reversal would involve a period of complete stoppage of ventilation, long enough to permit serious accumulations of gas.

2. In shaft- and slope-mines, at least, it is hardly practicable to have "every man out of the mine" during the night, and certainly impracticable to avoid keeping mules and horses inside.

3. The doors necessary to divide and direct the air-current are properly hung to open against the air-pressure; and, unless

* Received Feb. 14, 1910.

these were secured, a reversal of the fan would not result in a complete reversal of the ventilating-current, but only in a short-circuiting, with probabilities of accumulation of gas at many points. Moreover, there would be danger that important doors, blown open by the reversal, would remain open at the resumption of regular ventilation, thereby disarranging the entire circulation and inviting disastrous explosions.

The calculation under No. 3, in favor of this "standard" method, is based, as the author says (p. 404), on extreme figures; in fact, upon figures so extreme as to vitiate his conclusions. If old workings 5,000 by 5,000 ft. in area, 8 ft. thick, with 50 per cent. removed, have an open space of 100,000,000 cu. ft., they must be fully open, and could and should be properly ventilated; whereas, if such workings have closed, there would be no such open area, and, further, the movement of air or gas contained in them would be relatively very slow, and the "breathing" proposed would have a relatively small effect.

With regard to Mr. Stow's eighteen "arguments," the following comments appear to be warranted:

"A. AGAINST PLAIN PRESSURE-VENTILATION."

"1. *The traveling and haulage in the return-current, required by plain pressure-ventilation, are objectionable.*"

The discussion of this proposition is generally correct, except that locomotive-haulage is properly performed in a special split of air, and not in the general return; and that the difference in temperature between the fire-box of a locomotive and the electric arc is absolutely immaterial. Either will surely fire gas.

"2. *In winter there is serious risk, with plain pressure-ventilation, of the air-course freezing-up at a critical moment and possibly causing an explosion.*"

The question of freezing in the intake, especially in shaft-mines, is a serious one, and applies equally to both pressure- and exhaust-ventilation. I cannot agree to the proposition that the accumulation of ice induced in the hoisting-shaft during the night would be "merely a question of small expense and not of danger." Unfortunately, there have been a number of fatal accidents due to this cause.

"3. *Pressure-fans are objectionable on the score of possible breakdowns.*"

It is not clear that the small difference in pressure between exhaust- and pressure-ventilation has much effect in drawing the gases out of the solid; and I think the effect of the sudden breakdown of the only fan on a gaseous mine would be so serious that the type of ventilation in use would be insignificant in comparison.

Duplicate fans are certainly warranted in all cases on gaseous mines, and the insurance thereby secured should not be considered an "unnecessary burden." I see no reason why an exhaust-fan could not be reversed as well as a pressure-fan, at night or at any other time, should such a course be desirable.

"4. *Pressure-fans are more liable to be wrecked in case of explosions.*"

No fan should be so installed as to be liable to damage from explosion; and the relative damage to the two types would depend upon the location and course of the explosion, not on the type of the fan.

"B. IN FAVOR OF PLAIN PRESSURE-VENTILATION."

"5. *Positive or pressure-fans hold 'gas-blowers' in check better than exhaust-fans.*"

In my judgment, this argument is without importance.

"6. *Pressure-fans have the advantage in times of falling barometer.*"

This is not proved. The author suggests staying out of the mines until the barometer gets through falling; the safest course would be to stay out of the mines altogether. The final clause of his discussion would apply as well to suction-ventilation, and forms merely an argument against the driving of fans by constant-speed motors.

"7. *Positive or pressure-fans keep fires in check better than exhaust-fans.*"

The discussion is entirely correct; but the author might have added that reversing the current is often advantageously employed in fighting a mine-fire.

"8. *Under plain pressure-ventilation, when the air is properly split, an explosion should be less disastrous.*"

I would say that, under any type of ventilation, when the air is properly split an explosion should be less disastrous.

May we not assume that this paper is discussing properly-handled ventilation, which presupposes proper splitting?

"9. *In shallow mines, pressure-fans may force considerable volumes of gas to the surface through pillar-breaks.*"

This properly refers to mines worked to the crop, and not necessarily shallow. In such cases, the loss of air with pressure-ventilation may be very serious; and, for this condition, forms one of the strongest arguments against, not for, pressure-ventilation.

Nos. 10 and 11 have, in my judgment, no practical importance.

"C. Against Plain Exhaust-Ventilation."

"12. *Exhaust-fans necessitate, in winter, the daily removal of ice from the drift-mouth, or the hoisting-shaft, as the case may be.*"

This is fully referred to under argument No. 2.

"13. *Exhaust-fans, in winter, make the temperature uncomfortable at the head of the hoisting-shaft.*"

This is more than counterbalanced by the fresh air and freedom from smoke, gases, and dust at the foot of the hoisting-shaft, which should properly be put among the arguments for exhaust-ventilation.

"14. *In shallow mines, exhaust-fans may draw into the live workings considerable volumes of gas which would otherwise find their way, through pillar-breaks, to the surface.*"

Under these conditions exhaust-fans will draw in from the surface considerable volumes of fresh air, increasing the quantity in circulation, and decreasing the frictional resistance. Within limits, this is a decided advantage.

"15. *Exhaust-fans not only keep much the larger portion of the haulage-dust in the mine, but distribute it throughout the workings.*"

Since this argument received preponderating weight in Tables I. and IV., for bituminous mines, it is unfortunate that "lack of space" precluded the adequate discussion indicated. It would have been valuable to have had a classification of dust in respect to its origin and position; for it does not seem clear that the greater part is "haulage-dust." In fact, most dust-explosions seem to have started at the face, and it is at least possible that the dust due to blasting may have been a contributing cause. Should the blasting-dust prove to be the controlling factor, of course the argument is reversed to favor exhaust- instead of pressure-ventilation.

"D. IN FAVOR OF PLAIN EXHAUST-VENTILATION."

Except No. 16, to which the author properly gives no value, all arguments in favor of exhaust-ventilation, which (I may incidentally say) is the usual method in the anthracite-regions, appear only as arguments against pressure-ventilation; and a very important argument in favor of exhaust-ventilation is not referred to—namely, the presence of fresh and safe air at the foot of the hoisting-shaft or slope, giving a better chance for escape in case of accident or explosion, and permitting adequate lighting at this most important point.

Nos. 17 and 18.—Obstructed ventilation, involving the use of ventilating-doors or their equivalent at the drift- or shaft-mouth, while not fatally objectionable in drift-workings, presents, in shaft- or slope-hoisting, a problem so serious that no engineer would be warranted in using the method except as a temporary expedient.

Conservation of Natural Resources.

Discussion of the paper of James Douglas, p. 419.

JAMES DOUGLAS, New York, N. Y. (communication to the Secretary*):—In my paper on the Conservation of Natural Resources, I referred to the slow replacement of bee-hive ovens by the by-product ovens as a most notable instance of waste. And I quoted from Mr. Parker's report for 1906 an explanation given by him in accounting for the small production of by-product coke. It was that the market for the by-products of the coke-ovens was so limited that some of the ovens constructed were out of operation. His report on the manufacture of coke in 1908[1] does not record an improvement, and attributes the strange fact that we alone, of all the industrial peoples, delay the adoption of this cardinal improvement from the continuance of the same almost inexplicable cause. To quote again from his report, he says (p. 241):

"The year 1908 was not marked by any notable gain in the construction of by-product coking plants, though some new work was done. There was a net increase

* Received Feb. 2, 1910.

[1] *Mineral Resources of the United States for* 1908, Part II., U. S. Geological Survey (1909).

of 115 in the number of completed ovens in 1908 over 1907, the totals for the two years being, respectively, 3,892 and 4,007. The additional equipment consisted of 140 Koppers regenerative ovens built at Joliet, Ill., by the United States Steel Corporation, but this increase was partly offset by the dismantling of 25 Semet-Solvay ovens at Sharon, Pa., the net gain being 115 ovens. Included in the total of 4,007 completed ovens in 1908 are 152 Newton-Chambers ovens at Vintondale, Pa., but as no recovery of by-products was made at this plant in 1908, the production of coke is included with that from beehive ovens. The 56 ovens of the same type at Pocahontas, Va., have not been in practical operation since they were first installed. In addition to these there was one other by-product plant of 120 ovens that was not operated during the year. The number of retort ovens producing coke in 1908 was 3,679, as compared with 3,811 active ovens in 1907."

In describing the anomaly he says (p. 249):

"It has been contended that the development of the by-product coking industry would have shown more rapid progress if markets for the by-products were assured. This pertains essentially to the coal tar and its products, as there is no difficulty in disposing of the surplus gas, and there is practically at all times a fair demand for ammonia. As to the coal tar, the total value of this by-product from retort ovens at first hand in 1908 was $1,007,613. The value of the coal-tar products imported into this country in 1908, including duty paid, was $8,560,406. The values in all cases of imports are at point of shipment, and do not include ocean freights, commissions, and other expenses. It is probable that these importations have reached the consumer at a total cost of not less than $12,000,000, and in the three preceding years the cost probably reached $15,000,000."

These coal-tar products, however, which are imported into the United States at such a heavy figure, are all chemical extracts from coal-tar, such as salicylic acid, aniline dyes, and alkaline salts, the manufacture of which has passed in great measure into German hands. Some peculiar attribute of the German temper, and the thorough character of their technical educational methods, have given them a monopoly of this delicate branch of the chemical industry. Even England, where originated the manufacture of the coal-tar products, and where the first patents were taken out, has been unable to compete with her more precise and painstaking rival.

As the utilization of the tars is therefore the function of the chemical manufacturer, and the production of the crude material alone falls to the coke-maker, the one industry must keep pace with the other if progress along either line is to be made. As the profits of certain European coking-plants collecting the by-products are generally supposed to be from $0.75 to $1.25 per ton of coke on the by-products alone, it would seem as though capital, skill, and science could not be more profitably

employed in the United States than in removing this crying disgrace by turning the waste products from our coking establishments to such profitable use.

With regard to what will happen in the distant future when our coal-supply is exhausted, Dr. Robert Thomas Moore, in his presidential address[2] before the Institution of Mining Engineers in London on May 27, 1909, says (p. 455):

"Whether, indeed, it is a profitable matter to attempt to imagine the state of Britain 300 years after this, with its coal exhausted, or a world, say, 200 years later when it is all finished, is open to question. It is certainly beyond the scope or the objects of the Institution.

"I do not think it commends itself as an economic principle to restrict in any way the legitimate development of our mineral resources. They are a source of wealth to ourselves, and we are helping to develop the world. Is it not more reasonable to trust to the progress of science to discover some fresh method of utilizing the resources of nature to provide a substitute? Who would have expected, even 30 years ago, the immense possibilities for distributing light and heat and power that the development of electricity has opened up? We have the forces of the rainfall, the wind, and the tides to utilize to the utmost. We may even get our heat and power direct from the sun!

"Those who come after us have a long time in which to consider the problem, and we may safely leave it to them to solve in their own way.

"But that of which we should be careful is, that we should use our coal in the best possible manner—that in the working of it and in the using of it there should be no waste, either of men, of material, or of treasure; and it is the duty of an Institution such as ours to afford every aid to the presentation of any plan which will further the attainment of these objects."

His remarks upon the ever-increasing consumption of coal, despite the efforts of the engineer to economize, are worthy of quotation. He says (p. 453):

"It is a striking fact that notwithstanding all the improvements which have been introduced to economize coal in the various industries, the total consumption has gone on increasing. It seems as if the greater the economy becomes the larger is the consumption.

"There have been atmospheric engines, Watt's condensing engines, high-pressure engines, compound engines, triple- and quadruple-expansion engines, turbines, and gas-engines, each being an improvement on its predecessor, until the coal consumed per horsepower per hour has been reduced from over 10 pounds to $\frac{3}{4}$ pound; the methods of iron-smelting have been improved until the amount of fuel used has been reduced from 8 tons per ton of pig-iron to considerably under 2 tons; the processes for the manufacture of gas have been improved; and the whole history of the century has been a long series of savings in fuel. Yet the total consumption goes on steadily increasing. It would seem that the more the cost of power is cheapened, the more are the purposes for which it becomes available."

[2] *Transactions of the Institution of Mining Engineers*, vol. xxxvii. (1908–09).

Modern Progress in Mining and Metallurgy in the Western United States.

Discussion of the paper of D. W. Brunton, p. 543.

William Kent, New York, N. Y.:—The Institute may congratulate itself on the opportunity of reading the splendid address of President Brunton. It is an admirable summary of the progress that has been made in the mining and metallurgical arts in recent years. The paper rightly gives the chief credit for this progress to the aid which has been rendered to these arts by the advances in mechanics, chemistry, and electricity. It is notable that most of the headings in Mr. Brunton's address refer to the work of the mechanical engineer, the electrical engineer, and the chemist, rather than to that of the specialist in mining-operations. It seems that the most successful mining engineer of the present day is the one who makes the most intelligent application of the inventions and designs of engineers and others who are not directly engaged in mining. A tabulation of the headings of the address, grouped as below, shows this most clearly.

Work of the Mechanical Engineer—Rock-drills, hoisting-machinery, tramming, pumping, ventilation, dredging, sampling, concentration, briquetting, fume-recovery.

Work of the Electrical Engineer—Electric machinery for hoisting, pumping, tramming, lighting and signaling, electric transmission, electric smelting.

Work of the Chemist and Metallurgist—Explosives, chlorination, cyanidation, roasting, lead-smelting, and copper-smelting.

Work of the Geologist—Economic geology.

Work of the Mining Engineer—Mine-mapping, surface-mining, timbering, tunneling; and planning and supervision of the work in general.

The mining engineer of the present day, who is in responsible charge of a large plant, finds his chief work not in doing those things he was trained to do as a specialist, but in making use of the knowledge of geologists, chemists, and electrical and

mechanical engineers. He must be to a large extent not only an engineer, but an economist, a sociologist, and even a politician. I am glad to find that Mr. Brunton is all of these. On p. 549 he touches briefly the domain of politics: "While too much 'paternalism' is certainly to be avoided, it is doubtful if anything short of government regulation and inspection of explosives, detonators, and fuses will ever bring about the uniformity necessary to safety." In this statement he places himself on the side of the wise politicians of all parties. They are all opposed to "too much" paternalism as a general proposition; everybody is, except a few cranks, but we all favor "just enough" paternalism when it relates to particular cases in which we are interested. Every civilized government in the world is gradually growing more and more paternal, and the great body of the people is being benefited thereby. It required the paternal action of the United States government to bring about the "uniformity necessary to safety" in couplings on freight-cars, and it will require either State or national government action to bring similar uniformity in the use of explosives. We may decry paternalism in general, but in matters involving safety to life and health of miners and of wage-workers generally we can scarcely have too much of it.

On p. 560 Mr. Brunton touches a sociological, if not a political question. He says: "To-day we are beginning to realize that the public forms a third party, vitally concerned in the results of the work in which mining engineers are engaged." In this he strikes the keynote of our future national progress. The public is the great "third party" that is waking up to a knowledge not only of what it wants but of its power to get what it wants. One of the things it wants is a diminution of the death-rate in our mines, which now disgraces the United States. As Mr. Brunton says, "every one, no matter what his station, has a duty to society and to his fellow-men." The engineer's duty is clearly to do everything in his power to diminish the death-rate, not merely by his work as an engineer, but by his influence as a citizen in favor of at least as much "paternalism" as may be needed to safeguard the lives of our workmen.

Charles Catlett, Staunton, Va.:—It is to be regretted that Mr. Brunton did not include a chapter on cost-keeping and effi-

ciency-records. His wide experience would enable him to speak with equal authority on these subjects, and their proper development lies at the basis of success in the other departments on which he has touched.

Accounts should show what has been done, what is being done, and what can be done. In other words, they are a history of past and contemporaneous events, and are prophetic of events to come. Not uncommonly, accounts are confined to recording ancient history, so far past as to be of little value as a guide to successful operation, and make no attempt to point out the causes of the results noted. Even in this form they have value in telling the owners the loss or gain of the year, but may totally fail to fulfill their higher function of throwing light on the true efficiency of the management and of pointing out to what extent the work can be improved. A management may, by an unusual occurrence of circumstances, get results which are gratifying to the stockholders, and which add to its reputation, when its total efficiency is really very low.

We will never have thoroughly satisfactory accounts until another unit in addition to the dollar is used as a standard. The unit to be used is the "theoretically possible," and the actual results must be recorded in percentages of this possibility. These could be ultimately translated into the universal dollar. The costs in dollars and cents may be ignored for the time being, and there need be no fear that the highest efficiency, properly recorded in this way, will not also represent the maximum profit, if everything is taken into consideration. An astonishingly low cost in dollars or a disappointingly high cost in dollars may not tell us anything as to the true efficiency of the work. But if the attention of the management is day by day called to how far it falls below the theoretically possible it will inevitably lead to improvement.

This develops another suggestion. The weak point to-day is not apt to be the weakest point to-morrow, nor the weak point the day afterwards. One of the essential functions of good cost-keeping is that it should point out continually and automatically the weakest point, so that it can be corrected before it becomes chronic and before it has cost too much. A system of accounts which does this, and shows at the same time how far

each department or line of operation falls short of the possible, is a system which immediately encourages efficiency.

How quickly efficiency may be increased is illustrated by the common difficulty of ascertaining what a plant is actually doing, because the whole force do better work while they know the investigation is being made, even if there is not the slightest intention to deceive. A system of accounts which continually points out a man's more serious deficiencies, and encourages him by showing him promptly where he has improved, and how much, has the effect of keeping him up to a pitch which he has not reached before, and the "impossible" of one month becomes the "possible" of the next, and the normal of the third.

It may be contended that there is no fixed standard for the "theoretically possible" in any department, and that it can be determined only by investigation and experiment, and that it is different at different times. It is true; but the mere investigation and experiment to determine the "possible" at any particular period must increase the efficiency. From the percentage stand-point it makes little difference if the standard varies. The main question at any period is whether the efficiency is 50, or 75, or 90 per cent. of the possible.

It may also be contended that it is possible for a manager to fix on such a low standard as to make his record for efficiency higher. Of course it is, and it is possible for him to falsify his accounts, and one is about as likely as the other. The good men, and they carry the day in the end, know that they are doing honest, careful, conscientious work, and they will want to know the truth as nearly as possible. They will want to know what will help them, and are perfectly willing to stand on their records, and only want the facts in such form that they may be protected from improper criticism, which is certainly not true with the present method.

The market-values of railroad stocks have not decreased since uniform accounting has been required; rather the contrary. Neither would mining companies suffer by reason of uniform accounting. Nothing could be of greater service than a system of accounting which would carry the sanction of Mr. Brunton's approval, and which would be accepted, at least by all new mining companies, as a standard to follow, until gradu-

ally it would be universally adopted. Then the engineer who had to examine into a company would have a reasonable expectation of ascertaining within a moderate time those things which it ought to be the function of accounts to reveal.

Walter O. Snelling, Pittsburg, Pa. (This discussion is approved by the Director of the U. S. Geological Survey):—Mr. Brunton has succeeded admirably, in his discussion of recent progress in mining and metallurgy, not only in calling attention to the advances which have been made, but also to those directions in which the need for further improvement is most urgent. Particularly in his brief references to explosives he has touched many salient points, and as it happens that the U. S. government is already, through the Technologic Branch of the U. S. Geological Survey, making investigations along some of the lines which are mentioned, I believe a brief description of the work being carried on, and a statement of some of the results already accomplished, will be of interest.

In May, 1908, Congress authorized the establishment of a station for the investigation of accidents in coal-mines, and the work was taken up by the Technologic Branch of the U. S. Geological Survey, under the direction of Dr. Joseph A. Holmes. Pittsburg was selected as the location of the testing-gallery, both because of its position in almost the geographic center of the great coal-fields of Pennsylvania, Ohio, Indiana, and West Virginia, and because natural gas was available there which was of the same percentage composition as the average fire-damp found in coal-mines, thus enabling tests to be made in the presence of such gas mixtures as are encountered in actual mining-practice.

Even the most cursory examination of the causes of accidents in coal-mines shows that the improper use of explosives, and the use of improper explosives, is directly or indirectly to blame for a large percentage of the accidents which occur. Attention is sometimes called to the fact that statistics show that many more men are killed each year by falls of side and roof than are killed as the direct result of explosions, and from this fact the conclusion is drawn that the study of mine-accidents cannot greatly reduce the death-rate. This reasoning is most

faulty, for careful study not only shows that the many accidents which are the direct result of explosions can be in a large measure prevented, but also shows that falls of roof and sides are themselves very often the indirect result of the improper use of explosives; the constant firing of excessive charges, and the firing of shots "on the solid," being factors of great importance in so fissuring and breaking the surrounding strata as to make falls much more common than they would be if better mining-methods were employed, and if, by undercutting and by proper regulation of the explosive charge, the amount of the explosive was adjusted to the amount of work to be done.

As a result of preliminary investigations made with explosives, it was clearly shown that conditions very favorable to the production of mine-disasters can readily be brought about by certain types of explosive materials, and that different explosives vary greatly in regard to the readiness with which they ignite mixtures of fire-damp and air. Ordinary black blasting-powder, for example, is found to be able to ignite 8 per cent. mixtures of fire-damp and air, even when the quantity of powder used is as little as 25 g. As this quantity, somewhat less than an ounce, is, of course, too small to be of any value in mining-operations, it is evident that in a mine in which explosive mixtures of fire-damp and air are likely to be encountered, ordinary black powder is unsafe to use. Other explosives which were examined were found to vary in regard to their ability to ignite mixtures of fire-damp, and with several explosives many charges of 1,000 g. (somewhat over 2 lb.) have been fired without causing any ignition of the mixture of fire-damp and air. As soon as the preliminary tests had clearly shown the directions which the investigation of explosives must follow, a systematic series of tests were devised, intended to show the relative safety of all mining-explosives in the presence of such unfavorable conditions of gas and coal-dust as might be encountered in mines. In a testing-gallery, 100 ft. long, varying charges of each explosive are fired in the presence of mixtures of fire-damp and air, and mixtures of coal-dust and air, until the properties of each explosive in regard to the ignition of explosive mixtures are determined. By means of suitably constructed gauges, the volume of the gases and the pressure produced by the charges of each explosive are measured. The

length and the duration of the flame produced by the explosive are also registered upon a photographic film, and the rate of detonation, the heat evolved, and other physical and chemical qualities of the explosive are determined.

These investigations have shown that some explosives will ignite mixtures of gas and air more readily than they will ignite mixtures of coal-dust and air, while other explosives are more sensitive in regard to mixtures of coal-dust, and accordingly the standard tests have been arranged to show the relative safety of the different explosives when fired in mixtures of coal-dust and air, as well as in mixtures of fire-damp and air. Those explosives which pass all of the required tests are placed upon the "permissible" list, and it is a most gratifying fact that American manufacturers have already produced 29 explosives which have been found suitable to be so classified.

President Brunton refers to irregularities in composition of explosives as a possible cause of mine-accidents, and it is unquestionably true that any variation in the composition of an explosive, tending either to increase or decrease its strength or to change its properties in any way, may very readily be the cause of a mine-disaster. The miner gets a certain familiarity with the proper way to use the explosive, the proper charge to use, and the best manner of placing the hole; and when a change in the composition of the explosive is made, even though he loads the hole in exactly the same manner and with exactly the same charge, he may nevertheless obtain a blown-out shot.

To avoid this danger, one of the conditions required of an explosive, in order that it should remain upon the "permissible" list, is that no changes should be made in its chemical or physical nature, which might in any way alter its properties in regard to the ignition of explosive mixtures of fire-damp or coal-dust. Every explosive submitted for test by the Geological Survey is first subjected to careful chemical analysis, and such physical tests are also made as will establish its density, manner of packing, etc. If the explosive is found to answer all of the tests required, and is admitted to the "permissible" list, further samples of the explosive are taken from time to time from mines in which the explosive is being used, and these samples are transmitted to the explosives laboratory for analysis. As the samples are taken without previous notice, and from mines

where the explosive is in use, any variation in the composition of the explosive, or in its method of preparation or packing, which might alter its qualities in regard to the ignition of fire-damp, would be at once noted by the chemist. If such change in the chemical composition or the physical characteristics of any explosive occurs, the explosive is re-tested in the gallery, and does not remain upon the "permissible" list, unless the tests made show that the changes made in its manufacture have not altered its relative safety in the presence of mixtures of fire-damp and coal-dust.

As a result it is safe to say that the explosives which are on the "permissible" list will be found to be of uniform quality, and any variation of as much as 1 per cent. would be at once noted at the laboratory. I do not mean that a variation of 1 per cent. would necessarily require an explosive upon the "permissible" list to be again subjected to test, for there are many constituents in explosives which vary slightly in different mixes, and variations in these constituents do not necessarily bring about any change in the action of the explosive. But a variation of 1 or 2 per cent. in any ingredient of an explosive which has an important bearing upon the question of its safety in the presence of gas or dust, would at once necessitate a re-test. So far, then, as the explosives upon the "permissible" list are concerned, it may be said that the user is protected against any changes in composition or manufacture which might vary the safety of the powder in any way.

It is recognized by all users of explosives that the successful use of any type of dynamite or other high explosive depends in a large measure upon the selection of a detonator of the proper strength. When too weak a detonator is used the full strength of the explosive is never realized, and in this way it very often happens that a very large percentage of the strength of the explosive is lost. It is important, therefore, that detonators should be so classified as to enable the user to know at once the relative strength of different kinds.

This has not been the case in the past, and different manufacturers have selected wholly arbitrary methods of designating the strengths of the different grades which they manufacture; and even when the same manufacturer makes both electric detonators and fuse-detonators, or "blasting-caps," there has

not even been uniformity in regard to the naming of the strength of these two products. For example, detonators which are designated as "6 X," those called "No. 15," and those known as "double strength" electric detonators, are all of practically the same strength, and contain almost exactly the same charge.

In the course of the work with explosives a number of conferences have been held between the members of the Technologic Branch and the manufacturers of explosives. At one of these conferences the matter was brought up of the advisability of a uniform system of nomenclature in regard to detonators. It was pointed out that no advantage was to be gained by the existence of many arbitrary systems such as are at present in use, and that the users of explosives would undoubtedly be greatly benefited by the adoption of some standard method of naming the strengths of all electric and fuse detonators. The plan met with general acceptance, and I am pleased to be able to say that, as new printed matter is issued by the different companies, it is very probable that the new system of standard nomenclature will be followed.

The testing-station at Pittsburg has proceeded with its work in a very quiet way, and few people realize the great influence which it has already had in making conditions safer in coal-mining. I believe there are to-day a thousand miners living who, but for the work of the testing-station, would have come to their death in mine-explosions during the past year. Although this statement is simply an expression of personal opinion, it is not based entirely on surmise, but is a conclusion reached from personal contact with the very conditions that the testing-station is studying. Saturday of every week is a "visitor's day" at the station, and many of the coal companies, both those near at hand and those in other States, have brought their superintendents, mine-foremen, and many of their other employees, to the station to witness the tests. I have seen these men as they have watched the tests made with coal-dust which they had brought with them from their own mines, and have seen the expression on their faces as they saw the effects of a blown-out shot on this coal-dust in the gallery. As they see the resulting explosion, they realize that just such effects are possible in their own mines through a

similar charge of explosive, and they realize still more forcibly the fact that such an explosion as they can watch in safety at the testing-station would leave no witnesses if it took place within the colliery at home. Of course all these men have heard, time and time again, that a blown-out shot is dangerous, and that coal-dust will explode, but "seeing is believing," and when they go back to their work-places on their return from Pittsburg, I can assure you that the little heaps of coal-dust in the mine mean more to them than ever before. They can now realize the danger that lies in those little heaps of dust, and we have been told many times that rules made for the safety of the miners are never lived up to so closely as they are after a visit has been made to the station.

Miners are often careless, and it is generally the carelessness which comes from ignorance and indifference. When they realize that accidents are the result of definite conditions, when they see the ease with which these conditions can be brought about, and when they recognize the fact that in so many cases they and their fellows alone are responsible for similar conditions in their own mines, they seem to have a new understanding; and I believe the vivid image of the testing-gallery at the moment of a dust-explosion is not soon effaced from their memory, nor the realization of what such an explosion in the mine would mean to them and to their families.

The usefulness of explosives to man can scarcely be overestimated, and by their aid results can be attained which, by mechanical means alone, would be difficult or impossible. The great engineering-works of to-day are, in many cases, of such magnitude as to be directly dependent upon the use of explosives. But the interests of the public in regard to explosives are, I believe, of a somewhat different nature than they are in regard to any other commercial material, with the exception of foods and drugs. When a man buys fertilizer, for example, and the fertilizer is not of the proper strength, the man loses financially by the amount which the fertilizer is below the usual standard. In the same way with most commercial articles, if these materials are faulty in composition or construction, or impaired in strength, the buyer loses a certain percentage of the purchase price which he pays. But with explosives, the case is far more serious. If of faulty composition, the buyer not

only loses a certain portion of the strength of the explosive, but because of the irregularities in composition, he may even be buying, ignorantly, a most dangerous material, which will place his own life, and the lives of his employees, in jeopardy. That explosives of improper composition have been made and sold, and are to-day being made and sold, there can be no question. It is well enough to say that the buyer of explosives should familiarize himself with their composition, but I can assure you that this is a most difficult matter. The user of small amounts of explosive cannot afford the high charges which are made for even the simplest chemical examination, and even users of large amounts of explosives find that but few chemists are available whose experience with this particular class of bodies is sufficient to enable them to draw proper conclusions. And so I cannot fail to agree with Mr. Brunton in regard to his suggestion of some general and comprehensive supervision of explosives. Conditions which allow the manufacture and sale, without the slightest restriction whatever, of bodies possessing the properties of dynamite and nitro-glycerine, cannot possibly be for the general welfare, and I know of no field which offers greater need of proper and intelligent supervision.

Charles W. Goodale, Butte, Mont.:—The friction-system, designed for the recovery of flue-dust from furnace-gases, which was mentioned by Mr. Brunton as having been installed at Great Falls, has not been in use long enough to obtain a knowledge of its efficiency, but the following explanation of the system will be of interest.

Screening-tests on the blast-furnace charges of smelting-ore, and on the fine concentrate treated in the McDougall furnaces, show a high percentage of fine material, which would naturally be carried into the flues, and at a high velocity of the furnace-gases through the flues a considerable loss in dust would inevitably occur.

Before deciding upon the friction-system, a flue was built 300 ft. long, 4.5 ft. high, and 4 ft. wide, through which the furnace-gases could be drawn in measured volume and temperature, and at varying velocities. Two tests were made, maintaining a velocity of about 500 ft. per min., and with no obstructions or dust-arresters in the flue. The amount of dust

was determined per unit-volume of gas; then similar determinations were made with baffle-plates, Freudenberg plates, numerous contractions and expansions, and with wires. It was found that the wires gave nearly as high an efficiency of dust-recovery as the baffle-plates, and with very much less frictional resistance to the passage of the gases. It was then decided to build a dust-chamber of such dimensions that the furnace-gases would not have a velocity through it greater than 500 ft. per min., and to fill this chamber with steel wires.

All the furnace-gases from the several departments are assembled through individual flues in an uptake in the furnace-building. From the top of this uptake a cross-take leads over the smelter-buildings and tracks to the dust-chamber. The cross-take is 34 ft. wide and 20 ft. high. The main dust-chamber, rectangular in shape, is 367 ft. long, 176 ft. wide, 21 ft. high. Steel-wire netting, 1⅜-in. mesh, is bolted to the "I" beams of the roof, and at alternate intersections of the netting-wires, steel dust-arresting wires, No. 10 gauge, are hooked on, weighing about 1 lb. each and reaching nearly to the floor of the chamber. From the entrance to the chamber, and for a distance of about 150 ft., the space is fully occupied by wires, then comes a length of about 50 ft. with no wires, then 150 ft. filled with wires. In the vacant space, air-ducts, from both the basement and the roof, are provided, so that the temperature from that point on can be reduced and the condensation of arsenic effected on the wires. The purpose of this arrangement is to collect, as far as possible, the dust which is carried along mechanically by the furnace-gases in the first part of the chamber, which leaves the condensable elements to be recovered in the upper part. Experience has shown that at or below the condensing-temperature of arsenic, the wires become heavily coated, and it is therefore necessary to shake them. Provision has been made for this, but the arrangement can hardly be clearly described without a photograph or drawing. The dust-chamber is divided longitudinally by a partition-wall, and dampers placed at the lower and upper ends of the chamber make it possible to deflect all the gases through either half if it is so desired. Leading from the dust-chamber to the chimney, which is 506 ft. high and 50 ft. in diameter at the top, is a flue 1,200 ft. long, 48 ft. wide, and 21 ft. high.

In the floor of the dust-chamber there are more than 1,000 sheet-steel hoppers, arranged in 22 lines, and a complete system of tracks enables the dust to be drawn from any hopper at any time. The cross-take is also provided with hoppers, and a hopper-crane draws off the dust and conveys it to chutes leading down into the bottom of the uptake, where there are hoppers from which the material can be drawn into the charging-cars for the reverberatory furnaces.

There are 1,215,000 wires in the dust-chamber, weighing about 608 tons, and nearly 3,500 tons of structural steel in the flue-system.

Recording-thermometers and pressure-gauges have been placed at the entrance and exit of the dust-chamber and in the flue near the chimney, so that complete information regarding conditions will be available. It is my intention, when the flue-system has been in action long enough, to present a paper on this subject to the Institute for publication.

The use of wires as dust-arresters was patented in Germany by Rösing, who also took out a patent for it in the United States in 1890. The Freudenberg plates used in our experiments were of sheet-iron and were suspended parallel to the direction of the flow of the gases.

Ernest Levy, Rossland, B. C.:—Mr. Brunton has kindly asked me to give the meeting, as an illustration of that subdivision headed "Timbering" of his extremely interesting address, a short account of the methods used for filling excavations formed by ore mined from large masses.

I was managing for a time a mine situated on the largest known ore-mass in the world—namely, the San Dionisio lode of the Rio Tinto Copper Co., in the Province of Huelva, Spain. In order to give you a better understanding of the filling-methods employed, it will be necessary to go somewhat into other connected matters, as, for example, some of the characteristics of the ore-mass and the scheme of ore-extraction. The ore-mass, which reaches a length of about 1 km. and a maximum width of 220 m., has been developed more or less on the levels from the 7th to the 32d, which are separated by distances of 12.5 m. The development has in general taken the form of 4 by 4 m. headings, 10 m. from center to center, running

both with the length of the mass and across it at right angles to that direction, consequently, the ore left in place between each two levels may be thought of as a mass or slice of mineral with a thickness of 8.5 m. and an area equal to that of the lode at that horizon, supported on pillars which are 4 m. high, 6 m. square, and 10 m. from center to center. I am not sure if, on account of the huge size of the various masses belonging to the company, it was not originally considered that the small percentage of the mass thus extracted would be sufficient to supply all demands from these mines. However, whether this be so or not, I am of the opinion that the methods to be later employed in extracting the remainder of the ore had not been drawn up. The method later decided upon was to commence more thorough extraction by means of stopes running parallel with the width of the ore, approximately 100 m. apart and 15 m. wide. These stopes are divided into sections of from 30 to 40 m., according to the width of the mass at that place, in order that the rock-filling should not have to be carried an excessive distance from the stope-filling shafts. It is a characteristic of this mass that where the width is great the ore is soft and heavy, and where small, the mineral is extremely hard and stands well. Great care has to be exercised when removing the pillars in preparation for stoping, and also when excavating ground in the stopes themselves, that not too much weight has to be borne by the ground above the roof. The weight also becomes more effective when the natural fissures in the ore have become devoid of cementing material; these then form a plane of weakness. Water is a notable factor in effecting weakness along these planes by washing out soluble copper salts deposited in them at some former period. It is therefore a great necessity to catch up and remove from harm's way all water which may be found running into the mine, so that it can do no damage. A complete system of surface-drainage is resorted to in the neighborhood of the outcrop for the purpose of minimizing trouble from this source.

Timber, being scarce and expensive, is used as little as possible. As above mentioned, the spaces formed by removal of the ore are filled with rock in order to support the remaining portion of the mass. The rock is procured from the following

places: From the over-burden, hanging-, and foot-wall, rocks removed of necessity in preparation for and during open-casting, and from any rock afforded by underground workings. This latter source is only a very small one, since, as can be well understood, with such a huge and continuous mass but little dead work is called for. Naturally, it is of great importance that the rock used for filling should be of as close a texture and as strong as can be got, in order to be capable of supplying as nearly as possible the supporting quality of the mineral removed. The filling-rock is delivered into the mine partly through the main tunnel, which strikes the mass on the tenth level, and partly through main rock-shafts, through which it is taken to such levels as desirable. The rock is put down in mine-wagons, in which it is distributed from these main rock-shafts to interior rock-shafts, which are large enough to form deposits to be drawn from when required. The train-loads are dumped directly into interior rock-shafts. These interior rock-shafts have overflow-ways at such levels as are decided necessary, and the rock flows out on to a masonry platform locally known as a "submarine," about 2 m. higher than the floor of the level. The height of the submarine facilitates the shoveling of the rock direct into mine-wagons, and thence it is distributed where needed. Each stope-section has an individual rock-shaft leading from the floor above to the bottom of the stope, and through this is dumped all the rock used to fill that stope-section. Before stoping, all former excavations made, that is, the 4-m. headings, must be tightly packed, and in the filling there must be built main passage-ways, about 4 ft. wide by 6 ft. high, formed of side-walls and an arched roof, in such positions as to enable the ore to be mined and rock that is to be used to be distributed as required, and also as a means of ingress to pillars yet unremoved. Places for man-ways and stope rock-shafts in the walls of these arches are left where suitable, with a view to aiding future work. Arranging the plan for these arches, etc., is a matter which requires very careful consideration, in order that the scheme may be as effective as possible. At those places where it is decided to stope, it becomes necessary, first, to fill gradually (if this has not already been done) the spaces formed by the development-galleries, in such a manner that the pillars and those sections of the neighboring pillars lying within the confines of the vertical bound-

ing-planes of the stope can be removed. Here I may mention that the stope is so arranged that the center-line corresponds with the line passing through the centers of a string of pillars between the foot- and the hanging-walls. Consequently, before the stope can be started, there must be extracted the string of pillars together with the adjacent 2.5 m. of the string of pillars on either side. By degrees, all spaces thus formed are filled up tightly with the rock, and the first cut is ready to be taken from the stope. The ore is removed in horizontal sections of about 2 m. thick, as follows: A drift 2 m. wide is run in the first 2 m. thickness across the center of the section under consideration, and parallel to the length of the lode, which is parallel to the width of the stope. This connects the man-ways and the rock-shafts. The remainder of the ore from this layer of the section is removed in approximately 2 m. wide contiguous drifts; they start from the end of the preliminary drift, and run north and south, or parallel to the length of the stope, and extend to the ends of the section under consideration. Before each of these secondary parallel drifts is run, the roof of the previous one is supported by solid rock-filling. In placing the filling, a sufficiently strong retaining-wall is first built at a convenient distance from the solid rock, and then the intervening space is packed tightly with rock thrown in behind and stamped down.

Varying with the supporting-capacity of the ore, more or less ground can safely be left open while being worked; but, as a rule, all spaces are filled at the earliest possible moment. I think it will be understood from what I have said that the ore gradually becomes extracted and replaced by an equal volume of rock, and it is to be expected that some day a rock-filling placed by hand will stand in the place of a mass of ore which consisted of very many millions of tons.

W. L. Saunders, New York, N. Y.:—No one is more competent to discuss the subject of the modern conditions in mining and metallurgy than Mr. Brunton, for he not only speaks as one in authority, but his experience and his ability entitle him to a hearing as one of the first rank among mining engineers. The moral code set forth in the concluding paragraph of his paper is worthy to be placed as a classic in the annals of the

Institute, and it should form the basis of instruction to mining engineers at the colleges.

Under Section XI., Mine-Ventilation, Mr. Brunton says:

> "The ventilation and cooling of metal-mines have not yet received the attention which their importance demands. In this respect Western engineers could take profitable object-lessons from their brethren in the coal-fields."

The importance of this subject can scarcely be over-estimated. It should surely be the province of the mining engineer not only to excavate material and treat it properly and economically but in doing this he should study how to protect and conserve the lives of the miners. John Mitchell's figures show that four times as many men are killed in mines in the United States, in proportion to the number of men employed, as in any other country in the world. Explosions are responsible for much of this, but where explosions occur human life might be saved, provided there is a complete system of ventilation in the mine, and provided certain safeguards are employed. For instance, it has been urged by Mr. Mitchell that the introduction of compressed-air pipe-lines into all the workings of a mine might provide fresh air and even food to men imprisoned after explosions or through falls. This does not involve much expense, as mines are usually equipped with compressed-air apparatus, and the piping leading into the mine is of such a nature as to withstand considerable damage from the exterior. Furthermore, this piping at certain places, as, for instance, in the shaft, might be still further protected. Telephone wires inserted within the air-pipe might also serve a useful purpose in saving life.

Under Section VI., Underground Tramming, Mr. Brunton refers to the air- and electric locomotives which have come into general use, and he makes the statement, which no one can dispute, that each has its own field. Following this, however, the claim is made that "where the openings are dry and the roof sufficiently high and firm to carry the trolley-wire insulators, there is no question as to the desirability of using electricity." This seems to be a rather slender hook on which to hang the interests of the compressed-air locomotive. The members of this Institute recently visited the Anaconda smelters, where we saw air-locomotives doing useful service throughout the works. The superintendent, when asked why he used air in preference

to electricity, answered, because it was better and cheaper. This is only one notable instance where air is preferred for traction purposes. There are many others, as, for instance, the Homestake, the largest and richest single gold-mine in the world, where air-locomotives do useful service not only in the yards but in the mines themselves. These installations, Anaconda and Homestake, are of the old type; that is, the simple compressed-air locomotive. Notwithstanding this, the results are satisfactory and economical. There is a new type of locomotive, built by the H. K. Porter Co., of Pittsburg, which should be able to add 40 per cent. to the saving in air-economy. This new type uses the natural heat of the mine as a reheater to expand the air between the high- and the low-pressure cylinders. Under the old system of simple compressed-air locomotive it was frequently true that the fuel required to furnish the power for the air- and the electric systems was almost exactly the same. With the new system it is claimed that under the same conditions the fuel-requirement for the air-system will be but two-thirds of that needed with an electrical installation. There are some conditions where the electrical installation might prove more economical even than the compound compressed-air locomotive. Much depends upon the load-factor. With a good load-factor of from 30 to 40 per cent. of the rated power of the engines and generators furnishing the current, and with mining-conditions which permit operating locomotives at rated speed and power, it should be possible with electric locomotives to obtain efficiencies approximating those found in connection with the operation of large street-railways; but with ordinary mining-conditions there is much starting and stopping, tracks are crooked and curves frequent, and the ordinarily very poor load-factor results in an efficiency in mines even below that of the simple compressed-air locomotive.

Trolley-wires in mines are always more or less a source of danger, annoyance, and expense. This is especially true in gold-, silver-, and copper-mines, where many of the levels are operated simultaneously, and where the output per level is comparatively small. In such cases it frequently requires the services of several men and large quantities of copper-wire and insulators to keep the haulage-locomotives in close touch with the various working-places. Even in cases where the open-

ings are dry and the roof high and firm, the trolley-wire becomes a menace when there is a wreck on the road or any other accident resulting in a short circuit.

Wherever the wires are carried near ore-chutes, or places which require occasional or frequent blasting, they are in danger of being ruptured or put out of service. Furthermore, the danger from fire cannot be over-estimated, inflammable material being frequently in proximity to the wires.

The long entry to the mine can obviously best be equipped electrically, but in the various ramifications of the mine compressed air has been proved to be safest and best.

The argument that the air-locomotive loses time in charging has some merit, but observation of the performance of the electric locomotive in mines proves that the time lost in handling the trolley-pole under ordinary mining-conditions is approximately as great as that due to charging the air-locomotive. In narrow drifts it is sometimes found impossible to turn the trolley-pole, and the locomotive has to be run a considerable distance with the trolley-pole in advance of, instead of trailing behind, the support. Under such conditions very slow speed and great caution are required in order not to break the poles or tear down the wiring. Ordinarily, charging a compressed-air locomotive means the loss of about a minute and a half for every 4,000 ft. of travel.

It is to be regretted that Mr. Brunton dwelt so briefly upon the subject "Rock-Drills," for surely this valuable adjunct to the miner deserves serious consideration. The rock-drill has made the mining and smelting of low-grade ores profitable. Development-work, tunnel-construction, drifting, and stoping are all pursued to-day to a greater extent than in olden times, because the rock-drill has been perfected to that stage of simplicity where it may be used profitably and economically. It is difficult to find in the list of mechanical appliances a machine which has been subject to greater wear and tear, or in the building of which experience is of more importance, than the rock-drill. The design of a rock-drill is not by any means everything to be considered in looking for the best. Material and workmanship are of the greatest importance, and the skill which can only come of experience when applied in the construction of this important mining-tool should surely be of value

to the miner who is seeking a reliable machine for permanent service.

Mr. Brunton says in connection with the rock-drill that "a careful engineer will often have extreme difficulty in selecting the machine best adapted for a particular service." In discussing this point I am reminded of the story told of an American who was asked to attend a dinner in England and to make an address. He went fully prepared, but did not realize that English dinners are long-drawn-out affairs, and it was getting well beyond midnight when the Chairman announced the American's name, and said, "We shall now have his address." He arose and with a polite bow said that his address was No. 33 Queen Victoria Street and he would bid them all good-night. My address is No. 11 Broadway, New York, where I trust something may be done to relieve that "extreme difficulty" to which Mr. Brunton refers in selecting a rock-drill.

THOMAS KIDDIE, Northport, Wash.:—With reference to the subject of the deposition of flue-dust "by decreased velocity," referred to by Mr. Brunton, the members of the Institute who visited the smelter of the Tyee Copper Co., at Ladysmith, B. C., in 1905, were shown an experiment there being carried out on the lines of decreased velocity—first, for the purpose of lessening the amount of cold air drawn into the dust-chamber, and, second, for the better recovery of the flue-dust.

The draft was regulated by placing a damper in the stack and turning it down until the furnace-gases came out at the doors of the furnace; it was then raised sufficiently to draw all the gases away from the furnace at the lowest practicable velocity. The dimensions of the dust-chamber were 155 by 10 by 8 ft.

A comparison of the recovery of flue-dust for two periods each of two years shows:

First period, with a velocity of 1,225 ft. per min., gave 2.007 per cent. of flue-dust.

Second period, with a velocity of 440 ft. per min., gave 3.102 per cent. of flue-dust.

These results show that with decreased velocity there was an additional recovery of 54.50 per cent. The value of the flue-dust was $14.92 per ton.

A second plant has been similarly altered, with equally satisfactory results. In this case there were smelted copper-gold ores in which the gold-values were much higher than in the treatment at Ladysmith. Moreover, the size of the dust-chamber was much greater—namely, 300 by 16 by 10 ft. A comparison of the recovery for three years at high velocity with that of one year under decreased velocity gives:

First period (three years), with a velocity of 1,250 ft. per min., gave 2.91 per cent. of flue-dust.

Second period (one year), with a velocity of 450 ft. per min., gave 4.39 per cent. of flue-dust.

The additional recovery made under decreased velocity was, in this case, 50.8 per cent. During the latter period the higher saving in flue-dust was verified by the decrease in metal-losses. The value of the flue-dust was $30.20 per ton. In this second case, the flue-opening from the dust-chamber into the stack, situated midway between the bottom and top of the dust-chamber, was reduced in size from 16 by 10 ft. to 2.5 by 10 ft.

The Conservation of Coal in the United States.

Discussion of the paper of Edward W. Parker, p. 596.

W. L. SAUNDERS, New York, N. Y.:—Mr. Parker's paper, though entitled Conservation of Coal, might also be called the Conservation of Life in the Coal-Mines of the United States. No subject is of greater importance to mining-men at the present time than information from experts as to how to save coal and how to save human lives in mines. That the casualties in the coal-mines should exceed 3,000 in the year 1907 is simply appalling, and that 1,000 men should have been killed in a single year through explosions alone, and that so good an authority as Mr. Parker should say that "a prolific cause" is an "improperly-prepared blast" or "the failure on the part of the miner to undercut his coal," points to the importance of activity not only among mining engineers but also, through experts, by the legislatures of the respective States.

There is but little doubt that nearly all the serious coal-mine

explosions which have taken place in the United States during the past 10 years have been due to coal-dust alone, or coal-dust and gas mixed, and the ignition has been caused by blown-out shots; after very thorough investigation by State Mine Inspectors and Special State Commissioners, this has been fully proved in the large majority of instances. If, then, the blown-out shot is such a deadly agency, and the direct cause of the death of so many thousands of coal-miners, it is natural to ask whether or not such things are preventable, and if they are preventable, how? If the hole is drilled in the proper place to the proper depth, charged with the right amount of powder, and properly tamped or stemmed, a blown-out shot is an impossibility, providing, of course, the coal has been properly prepared for blasting. If then the application of ordinary knowledge, of ordinary skill and experience would eliminate this frightful danger, why is it not done? Because the modern mine-crews are largely made up of men who are inexperienced, unskillful, and densely ignorant. These men are allowed to drill their own holes; to charge and fire them, notwithstanding that the lives of all the men in the mine are depending on the good judgment of each individual man. At many places all the men have to be out of the mine before the shots are fired, and this dangerous work is performed by shot-firers. In the State of Illinois there is a law which compels this precautionary measure. How dangerous this occupation is may be inferred from the fact that during 1907 12 shot-firers lost their lives by explosions in Illinois. Had the miners been at work when these explosions occurred the loss of life would have been frightful.

Illinois produces more than 40,000,000 tons of coal per annum, and in 1907, out of a little more than 40,000,000 tons mined, nearly 25,000,000 tons were blasted from the solid; the bill for powder amounted to $2,208,343, and represented 1,261,910 kegs, almost enough to make one suspect that the coal-operators of that State are in league with the powder manufacturers of the United States.

It is interesting to notice that where the coal was undermined by machinery, each keg of powder produced 96.02 tons of coal, while from the solid each keg of powder blasted only 25.78 tons. This comparison needs no comment.

Perhaps the simplest description of a blown-out shot is one

that does no useful work in shattering or blowing-down the coal, but blows out its tamping and projects a long, vivid tongue of flame into the chamber where it is fired, the floor and sides of which are usually covered with coal-dust; this coal-dust is raised in clouds by the concussion, and in this diffused condition is easily ignited, and an explosion occurs, which goes through the mine with inconceivable rapidity, carrying with it death and destruction of property, the extent and violence depending on the amount of dust and good air (oxygen) in the mine. If 1 or 2 per cent. of fire-damp (CH_4) is present in the air, the dust ignites more rapidly. Until recently it was a disputed question as to whether coal-dust could be exploded in the absence of fire-damp, but this question has been settled beyond controversy by the physical tests which have been made, both in the United States and in all the coal-mining countries of Europe, in which thousands of people have seen coal-dust without any admixture of gas exploded by using a cannon-shot to represent the blown-out shot, and an iron tube 100 ft. long and 6 ft. in diameter to represent a mine-gallery; coal-dust is strewn on the bottom and on shelves along the side, the cannon loaded with black powder and stemmed with fire-clay is fired into the tube, and a terrific explosion occurs. This has been done hundreds of times and settles the question for all time.

It has also been fully demonstrated by this same method that there are many explosives which do not ignite coal-dust as readily as black powder does, on account of the very much shorter flame produced, and these are consequently much safer to use in mines; but as no explosive is flameless, this, while it will prove a mitigation, does not promise perfect immunity by any means.

A blown-out shot occurs when the tamping is the path of the least resistance—this is likely to happen when the hole is drilled beyond the undermining, or when it is drilled on the rib, away from the undermining, or where blasting from the solid is practiced entirely. In some cases it is the result of carelessness; in many the result of ignorance; in some of neither, but of improper preparation of the coal. In thick seams of coal, where the undermining is done with chain-machines, where the undercut is 6 or 7 ft. deep and 4 in. high, the hole is generally drilled in the rib at such an inclination that it will touch or nearly touch the roof at the back. In this case the coal is not prop-

erly prepared; it is almost as bad as blasting off the solid, in some cases worse, for the way the hole is drilled makes it more dangerous. Many of the most disastrous explosions have occurred exactly under these conditions. It is a significant fact that the most destructive colliery-explosions which have occurred in recent years have been caused by blown-out shots, where the coal has been mined with chain-machines. Harwick, Monongah, Darr, and Marianna are notable examples: in these four explosions nearly 1,000 lives were lost.

The truth is, that undermining coal is not a sufficient preparation for blasting; it should be sheared on one side or in the middle; in this condition less than one-half the powder would be necessary, less than one-half of the smoke would result, giving better sanitary conditions, and less danger of falling roof, which is largely caused by excessive blasting, and is a prolific cause of injury and death. All the coal would be in a better condition for handling and rehandling, giving very much less slack at destination, and a positive gain of at least 20 per cent. more lump, and, in addition to all this, immunity from blown-out shots, from destruction of life and property.

The value of shearing in the preparation of coal is recognized by mining engineers, mine-superintendents, and mine-foremen, as well as coal-operators, but because they think it adds to the cost of production, they are willing to forego all the many advantages which it gives. It would add little, if any, to the cost of coal, for when lump coal is worth from $1 to $1.40 per ton and slack coal is worth from 20 to 40 cents per ton at the mines, and shearing will increase the more valuable coal by 20 per cent., and the cost of shearing with approved shearing-machines would be about 6 cents per ton, it is not hard to figure what it would cost.

Shearing will be universal some day, and it will be a blessing to the workman and to the operator, and to all interested in the conservation of our national fuel-supply.

It is likely that in a comparatively few years the vast bulk of our bituminous coal will be produced without explosives of any kind; it will be excavated and loaded by mechanical means entirely, reducing explosives to a disappearing minimum, and so reducing accidents by falls of roof that a death through this cause will be as rare as it is now frequent.

A great many more deaths result from falling roof and coal than from all other causes combined, but little notice is taken of them because the fatalities occur only one or two at a time, and are scattered all over the coal-regions, wherever there are coal-mines in operation. This great loss and waste of life should not be less appalling because there is nothing spectacular about it—the sorrow and suffering are just as acute, there are just as many widows and orphans, just as many bereaved fathers and mothers, as if the lives had been lost in explosions, and just as earnest efforts should be made to prevent deaths from this cause as from any other. It is easy to say that in most cases it is carelessness or ignorance, or both, but this does not relieve us from responsibility, especially if ignorance is the cause, for if we employ workmen who are ignorant it is our duty to teach them, and so to safeguard their lives and others' that they shall not lose them through ignorance; and much can be done in this way, and hundreds of lives saved every year, materially reducing the disgraceful list of deaths through this cause. Fully 95 per cent. of all fatal and non-fatal accidents caused by falling slate or coal take place in the working-faces where the miners are engaged in digging or loading the coal. In some mines the roof is naturally bad, and should be carefully and systematically timbered, the props having over them stout, broad cap-pieces, presenting a wide surface to the roof and set as near the face as necessity may demand. If this is done the miner can work between the props with perfect safety; but as he looks on this work as being unproductive, he will not take the time, or he may feel indolent and neglect to make his working-place safe. In order to overcome this, there must be stringent rules compelling systematic timbering after an approved method, which would make all working-places safe; it might be objected that some places do not need as much timbering as others, but the answer to this is that where there is such ignorance, all places must be looked upon as dangerous. One of the most prolific causes of falls of rock and slate in the faces is blasting, especially where the coal is undermined by chain-machines, or blasting off the solid is practiced, as exceptionally large shots are necessary; these shots jar the roof to such an extent that the roof, which before was comparatively safe, becomes loose and dangerous, and too often the miner goes into a place

which has been thus rendered unsafe and commences to load coal under a roof which may come down, and frequently does come down, maiming him or crushing out his life. At least 75 per cent. of these accidents can be avoided if the mine-managements adopt proper methods which are rigidly enforced. As already indicated, one method is in systematic timbering; if in addition to this all blasting from the solid is made a criminal offense, and blasting is permitted only when the coal has been undermined and sheared on one side, then the roof would sustain but little, if any, injury from powder-shocks. If this precaution were observed, and a man were employed to visit each place once (or, if necessary, twice) a day, making sure that all the rules are enforced, deaths by falling roof would be rare, instead of daily, occurrences. In the North of England one man, who is called a Deputy, is employed for every 20 or 30 men. His business is to look after their safety, to visit every place as often as necessary, each shift, to see that each place is amply supplied with timber of proper length, and in cases of peculiar danger to set it himself; he carries an axe and saw, and is always ready for emergencies. These miners are to the manner born; they come of long generations of miners, and in all the world there are none more skillful or intelligent, all speaking a common language. If it is necessary to throw such safeguards around them, how much more necessary it is where we have such a lack of skill, such dense ignorance, and so much difficulty in oral communication.

Dust-Explosions in Coal-Mines.

Discussion of the paper of Franklin Bache, p. 667.

R. W. Raymond, New York, N. Y.:—I think Mr. Bache has put his finger on the chief source of the danger of dust-, or gas-and-dust, explosions in collieries. I mean the persistent determination of the miners' unions to increase their weekly wages by the excessive use of explosives. This would not be feasible if coal-miners were paid by the day; but this form of payment is, for many reasons, not economically practicable; and the universal practice is to pay for the winning of coal according to the quantity produced. If the miner, by using a large amount of powder, can throw down a large amount of coal without corresponding labor on his own part in undercutting and drilling, he will receive more money for less work, provided he is paid for everything—merchantable coal, worthless dust, slate, and "bone"—resulting from such a method.

My attention was called to this matter many years ago by an admirable report of Prof. W. B. Potter, a past-President of the Institute, on the conditions obtaining in this respect in the Illinois coal-field. It was made very clear in that report that considerations of danger to workmen or loyalty to employers could not be relied upon to prevent miners from employing this perilous and wasteful method of increasing their own immediate receipts.

So far as I know, only three remedies have been attempted, namely: the enforcement of discipline as to the methods of mining; the refusal to pay for dust, etc., produced by the miners' methods; and restriction upon the use of explosives, effected by requiring the miner to purchase them from the employer, at a price so high as to make it unprofitable for him to use them in excess. All of these attempted remedies have encountered the bitter opposition of the miners' unions. The enforcement of discipline has become almost impossible, if discipline be (as it should be) understood to involve punishment for the violation of rules when no disaster has followed. I

know of a comparatively recent case in which a miner, in a highly "fiery" colliery, was found to have matches in his pocket, contrary to express rule. But the superintendent did not dare to discharge him, knowing that a strike would immediately follow, and that, even if the committee of the Union should ultimately decide that the discharge was justifiable, and should order the men back to work, there would be a delay of one or two weeks in the operation of the mine, the cost and loss of which would fall upon the company. A week or two later there was a fearful explosion in the same colliery, destroying many lives. And I saw afterwards, in a respectable journal, the statement that this loss of life was chargeable to "the greed of capital."

The second remedy—namely, the refusal to pay the miner for worthless material, has been a fruitful source of trouble. It was hardly to be expected that the miners' unions, maintaining an attitude of war towards employers, and regarding every interval of peace as simply an armed truce, would regard as just the deductions made by officials from their car-loads of miscellaneous stuff. Though forced to recede from their original demand to be paid for everything, they still protested against unfair treatment; and this system, it must be confessed, left room for such complaints.

The third remedy, adopted after much consideration and experiment in the anthracite-regions, proved the best of all. The miner was required to buy explosives at a price (higher than the market-price) which made it unprofitable for him to substitute powder for labor. At the same time, the extra cost of the powder which he did use was taken into account in fixing his remuneration, so that he remained, after all deductions, the best paid of all laborers of his class.

Unfortunately, the true meaning of this arrangement seems not to have been brought clearly to the attention of the "Roosevelt anthracite commission." It was widely misrepresented as an attempt on the part of the operators to squeeze out of their employees a miserable extra profit; and the operators, anxious above all to avoid the continuance of disastrous conflict and the weight of popular odium, surrendered the point without adequate defense. So the Commission annihilated by its report the result of many years' study of the

subject, and the most acceptable and effective automatic remedy for a great evil—a remedy which had operated successfully for years before it was thus summarily discredited. I fear it will be long before we succeed (if we ever do succeed) in restoring an arrangement so satisfactory.

Mr. Bache points out another illustration of the way in which public sympathy is utilized for special interests. I refer to the statement in his paper (p. 670), that mining companies and State legislatures are influenced by humane considerations to provide that blasts shall be fired by persons specially appointed—in other words, that the miner who has prepared a dangerous shot shall be forbidden to fire it himself! There is reason to believe that this provision has been utilized in some instances for purposes of private revenge—the hole being bored to extra depth and overloaded and the fuse so arranged as to "get" the shot-firer, against whom the miner had a grudge. A simple remedy for this evil would be to ordain that the man who fires the shot should load the hole.

The official Federal investigations as to the possibility and the peril of explosions and mine-fires due to the presence of coal-dust have confirmed similar investigations abroad, and given us much interesting information about our own conditions. But they will not develop the chief danger until they expose fearlessly the causes which Mr. Bache has indicated.

Borax-Deposits of the United States.

Discussion of the paper of Charles R. Keyes, p. 674.

A. M. Strong, Bishop, Cal. (communication to the Secretary*) :—The paper of Mr. Keyes gives us the most complete account of the geology of the borax-deposits in the Death Valley region that has yet been published, and is a valuable addition to the geological literature of a very interesting though little-known territory. During the past ten years I have been over much of the northern part of the country described by Mr. Keyes and the region immediately north of it.

* Received Nov. 1, 1909.

Mr. Keyes seems to give the impression that Furnace creek is the northern end of these borate-bearing deposits, but such is not the case. Of the seven principal marsh-deposits which for years furnished the greater part of the borax-supply in the manner described by him, four are north of the boundary-line as given on his map. The entire territory immediately east of the Sierra Nevada is composed of a series of sinks, each with a well-defined drainage-area. Of these, Death valley is the deepest, and its drainage-area one of the greatest. The section given in the paper is typical of any line drawn eastward from the Sierra Nevada. At the lowest point of each of these sinks is a marsh, which, in times of heavy rain-fall in the mountains, is converted into a lake. These marshes always contain borax, which is now considered to have been derived from the washing of the Tertiary lake-beds in the surrounding mountains, of which they form a considerable part. Additional territory which I have studied shows that the boundaries given on Mr. Keyes's map should be considerably extended to the north. Just how far these borate-beds extend I am not prepared to say. J. E. Spurr says of them:[1]

". . . these older colemanite beds are characteristic of the broad belt of earlier Tertiary sediments which runs northwest and southeast in the region lying immediately east of the Sierra Nevada and which reaches at least as far north as northern Nevada and as far south as the Mojave desert."

In the Tertiary sediments north of those outlined by Mr. Keyes, colemanite has been reported by W. H. Storms and others in Fish Lake valley, and I have found it within the Saline Valley drainage, although in neither place is it in paying quantities.

In the main, I agree with Mr. Keyes in regard to the general geology of the country, though there is not much data, as yet, on which to work, but I take exception to his statement (page 694):

"The borate-bearing deposits are usually spoken of as lake-beds. Upon what grounds I do not know. Lithologically, they appear to be the same from Death valley to the Pacific ocean. Only in the western part of the Mojave plain have fossils been found, and these are marine Eocene and Miocene types. It seems probable that if strictly marine beds extend this far from the Pacific into the Mojave area, the Death Valley beds are also deposits of the sea rather than of extensive lakes in the process of desiccation."

---

[1] *Professional Paper No.* 55, *U. S. Geological Survey,* p. 161 (1906).

I consider that there is ample data on record to identify these deposits as fresh-water lake-beds.

Mr. Spurr says:[2]

"The Tertiaries at Silver Peak contain remains of fresh-water mollusks, fish, and plants, as well as coal beds."

He then lists a number of specimens from within the Fish Lake Valley drainage-area. Later he says:[3]

"Farther south, in the Funeral Range, at Furnace Creek, the writer observed upturned Tertiary strata resembling those at Silver Peak. The series is several thousand feet thick and rests unconformably upon Paleozoic limestones. The beds are conglomerates, clays, sandstones, and chemically precipitated limestones or travertine, which also occurs in the Tertiaries at Silver Peak. The Furnace Creek beds are often gypsiferous, contain grass remains, and include also a bed of borate of lime, or colemanite. No fossils have as yet been found in them. These Tertiaries extend southward and are widely distributed. In and near the El Paso Range Mr. H. W. Fairbanks has described a series of tilted clays, sandstones, volcanic tuffs, and interbedded lavas. This series is at least 1,000 feet thick. A seam of coal occurs in these beds, near which are found fossil leaves which Doctor Knowlton regarded as probably Eocene. Farther south, in the Mojave Desert, near Daggett, is a folded and faulted series of sandstones, shales, and tuffs more than 1,000 feet thick. The series overlies rhyolite and contains a bed of colemanite. Mr. W. H. Storms who has described this occurrence, regards the period of folding as perhaps Oligocene."

Sydney H. Ball, in speaking of this territory, says:[4]

"The Eocene inaugurated the Tertiary period of volcanism and lake sedimentation processes, accompanied by important deformation, erosion, and ore formation."

Also:

"The climate must have been moist and the presence of fossilized wood in the lake beds shows that trees flourished near its shores. While the lake was thus for the most part fresh, periods of aridity alternated with those of comparative humidity, and the lake or portions of it were partially desiccated, permitting the local precipitation of limestone, gypsum, and boron minerals."

Again, in speaking of Furnace creek, he says:[5]

"The Tertiary lake beds consist of white, yellow, and green consolidated clays, friable sandstones with ironstone concretions, rounded and subangular gravels, and

---

[2] *Professional Paper No.* 55, *U. S. Geological Survey*, p. 20 (1906).
[3] *Op. cit.*, p. 21.
[4] *Bulletin No.* 308, *U. S. Geological Survey*, pp. 40, 41 (1907).
[5] *Op. cit.*, p. 198.

thin limestone lenses. Much of the clay shows sun cracks and ripple marks, indicating that the lake was at times shallow and even dry. The subangular form of certain of the gravels indicates that cloudbursts at times spread sheets of detrital deposits over the lake beds. Colemanite and other chemical precipitates interbedded with the other deposits were laid down during periods of unusual evaporation."

Statements of the same kind can be found in all of the publications of the U. S. Geological Survey relating to this territory. On the western slope of the White mountains and Inyo mountains are thick beds of clay containing large numbers of fresh-water mollusks and quite a percentage of phosphates. Mastodon remains have been found in the Owens River valley, and this summer I found similar remains in a clay-bank under the lava cap on the west side of the Panamint valley; also land-snail shells in the surface-soil of a granite hill-side a short distance away.

The geologic history of this territory since the end of the Jurassic period is very different from that of any other part of the United States. During part of the time the Sierra Nevada was covered with an immense glacial sheet. In the passing of this sheet much of the water flowed eastward, as shown by the moraines in all the cañons on the eastern slope. This was associated with a period of intense volcanic activity, with its attendant elevation and depression of ranges and valleys, making constantly-changing drainage-areas and occasioning a great amount of erosion of recently-formed lavas. That the water area was large and constantly changing is shown by the great extent of the lake-bed deposits and the presence of the remnants of old river-channels now high on the ranges. The remains of the animal and plant life, so far as known, show that the climate was moist and warm, at least at times. These conditions resulted in the accumulation of considerable amounts of boron salts and associated minerals in the water of the lakes. A cutting-off of portions of these lakes by fresh lava-flows or other disturbances, and their resulting desiccation, left the colemanite-deposits at different points in the general stratification of the sedimentary rocks. The presence of numerous small sinks, some with a drainage-area of only a few acres, within the drainage-areas of the large sinks, gives some idea of the geological changes of the past. Mono and Owens lakes, including in their drainage-areas the well-watered

eastern slope of the Sierra Nevada, are good-sized bodies of water, kept up by an inflow equal to the evaporation. Still, they have all the characteristics of the marshes of the other sinks. Both contain a considerable percentage of boron salts, and the water of Owens lake is extensively worked for the soda it contains. It is quite probable that until quite recent times there was a general connection from this territory to the open sea by way of the Santa Clara valley, as suggested by Mr. Keyes.

It will be slow and difficult work to get at the relationship of the different strata and exposures of the lake-beds. Constantly-changing conditions during the times in which they were deposited, the great faulting and tilting that they have undergone, the desert country in which they are located, and the roughness of the ranges dividing the existing drainage-areas, all add to the complications. Still, they are of great interest to the prospector as well as to the geologist.

Colemanite may be found in other places than those now known; all the soluble salts of the alkalies are found, and it is quite probable that many of them will be obtained in large enough quantities of a purity to make them of commercial value. Several small beds of sulphur are known. Gold- and silver-deposits are found in the Tertiary andesites and rhyolites. Some of the old fragments of river-channels have been found to contain placer gold, and I found one place in which the clay of the lake-bed deposits contained an appreciable quantity of gold. Much remains to be done before a true idea of the geological and commercial condition of these Tertiary deposits can be formed.

Glass Mine-Models.

Discussion of the paper of Edmund D. North, p. 755.

A. Scott Reid, London, Eng. (communication to the Secretary*) :—As a constructor of several glass mine-models, I have read with much interest the description of the model of the Montana-Tonopah mine-workings by Edmund D. North.

---

* Received Feb. 25, 1910.

As an interchange of ideas and experience is always an advantage, and in view of the fact that such models are coming more and more into vogue, perhaps the following remarks on this subject may prove of interest to mining engineers generally and to Mr. North particularly.

In the model of the Waihi gold-mine, New Zealand, which I constructed, a dust-proof case is used. The top, bottom, front, and two ends are of glass; the back only is of wood, painted a dull white, against which the mine-workings show very distinctly. Each mine-level is represented by a horizontal sheet of glass, supported on metal runners fixed to the uprights at each end of the case, in a manner similar to that described by Mr. North. The corner uprights are strengthened by having a metal band screwed to the inside face of each, to which the metal runners or supports for the sheets of glass are screwed in turn. The reason for having this additional strength is that as the mine is developed in depth, and fresh levels are opened, the increasing weight of the glass sheets, added to the model, caused the unsupported wooden uprights to bend slightly, so that the glass sheets did not fit in with the desired nicety, and co-ordination was, to a considerable extent, lost. Strengthening the uprights in this manner is not important in models of mines in which only a few levels have been opened, or when the sheets of glass are small, but in the Waihi model there are 15 sheets of glass to be supported, and more to be added as development proceeds, and the weight is considerable. The glass sheets are 24 by 40 in. in area by 0.25 in. thick. Thick glass is used because thinner sheets of this large size sagged somewhat. The front only of the model is removable, and is held in place at the bottom by metal pegs fitting into holes in the base stretcher, as in Mr. North's model, and by key-locks at the top.

The mine-workings are traced on the glass from the plans in a manner similar to that described by Mr. North. Ordinary artist oil-colors are used, reduced to a workable consistency by diluting with "megilp" or turpentine; also a little prepared sugar of lead is added to hasten the drying. I have found a ruling-pen to be the best instrument with which to do the lettering on the glass; the other work is done with a small sable-hair brush.

The idea of using a different color for the drives and cross-cuts at each level is good, and must be of great advantage in many cases. In the Waihi model all the drives and cross-cuts at each level are drawn in black, and the reef is painted on in crimson, the walls being indicated by a thin vermilion line. The width of the reef is indicated by vermilion figures at the ends of the various cross-cuts through the reef and at other desired points. All figures, signs, and notes in red refer to the reef; all in black refer to levels, number of levels, the depth down the shaft, names of cross-cuts, etc., other colors being used to indicate geological features.

Another point of difference between Mr. North's model and those which I have constructed is that I use thin sheets of gelatine instead of glass to represent the vertical sections. This gelatine, which is made in sheets, 23 by 19 in. in size, is cut into strips just sufficiently wide to clear vertically the space between two levels, and when set in position the strips represent the standing reef between two levels. The sheets are placed in position along the reef on the glass, following the windings of the reef, and are held in place by means of small angle-pieces of the same material, which are fixed to the glass and to the vertical gelatine strips by means of transparent glue or cement. The strips of gelatine are, of course, sloped to correspond with the dip of the reef between levels, the term "vertical" being only approximate, since the gelatine strips will be vertical only when the reef is so. The gelatine sheet is easily manipulated and cut with scissors, and, if held in a moderate heat, it can be bent to the desired curve, which it retains when cold. The angle-pieces are made by holding small oblong pieces of the gelatine against an electric lamp and folding them to the required angle. On these vertical gelatine strips the stopes and winzes are shown. As stoping proceeds, the gelatine is gradually covered with crimson paint until the whole of the ore between any two levels is taken out. When necessary to indicate the thickness of a reef between two levels, this is done by sticking on pieces of gelatine transversely to the strike of the reef, and painting on these pieces the cross-section of the reef. The vertical glass cross-sections in Mr. North's model serve this purpose, and, as he points out, at the same time act as supports to the horizontal glass sheets.

Winzes are shown by broad black lines drawn on the gelatine, and where these are sunk in ore a narrow strip of crimson paint is drawn down each side; these crimson lines are omitted when the winze is in country-rock or in unpayable quartz. The gelatine strip is added to, from winze to winze, as the stopes are extended longitudinally. Obviously there is no need to put in any gelatine where the reef is unpayable and will not be stoped.

The contour of the surface of the ground is indicated in a general manner by vertical strips of gelatine, radiating in all directions from the highest point to the boundary-lines of the mine. These strips are cut along the top edge to the required slope to represent the hill in which the upper workings of the mine are situated, and a green line about $\frac{1}{8}$ in. broad is painted along the top edge. As the ground slopes away the green line is continued on the gelatine on successively lower glass sheets.

Pillars of wood, $\frac{3}{16}$ by $\frac{6}{16}$ in. in cross-section, painted black, represent the hauling- and pumping-shafts, and $\frac{3}{16}$ in. square pillars represent filling-shafts.

The top and bottom glass sheets are ruled off into 2-in. (= 100-ft.) squares, lettered alphabetically along the top and bottom, and numbered along the ends. The boundary-lines are drawn on each level; and the bottom glass, instead of being ground, as in Mr. North's model, has strained over it a sheet of thin tracing-drawing paper, which is rather more opaque than ordinary tracing-paper, and diffuses the light from the electric lamps underneath very satisfactorily. The model stands on supports 8 in. high, and in the space between the floor and the glass bottom of the case the electric lamps are placed. These lamps, six in number, are fixed in groups of two on three oblong wooden blocks which lie unfixed on the floor; the blocks measure 10 by 3.5 by 1 in., and are connected with each other by 18 in. of loose wire. The object in having the lights movable is that the whole six lights may be concentrated when desired, but normally they are spread out to give the best general light to the model.

The scale of the model is 50 ft. to 1 in. Two other models, similar in all respects to the Waihi model, except that they are 4 in. shorter, are necessary to contain the Waihi mine-workings. The total height of the model, from the floor to the top of the case, is 3 ft. 9 in.

The Cyaniding of Silver-Ores in Mexico.

Discussion of the paper of Albert F. J. Bordeaux, p. 764.

HERBERT A. MEGRAW, San Luis de la Paz, Guanajuato, Mex. (communication to the Secretary*) :—Although Mr. Bordeaux prefaces his paper with the statement that it is a general outline of practice in the Republic, many of his statements seem to be made from the view-point of a particular case, which does not always agree with general practice. For the purpose of calling attention to some of these variations, I respectfully offer the following comments:

The operation of crushing and grinding is performed by so many different methods and systems that it is useless to select any one as the standard. However, primary crushing in rock-breakers followed by stamps is thus far so general that it might be so classified. Whether or not it is the best way, is open to question. Regrinding may be performed by any of the various types of Chilean mills, Huntington mills, or similar machines, or the material may be taken directly from the stamps and delivered to the tube-mills.

Mr. Bordeaux's statement that "the slimes can be treated only by decantation; the new Butters slime-filter, as used in Nevada, does not seem to be successful with silver-ores," is surely an error. All over Mexico many of the purely silver-ores are being successfully handled by the Butters or similar filters, such as the Moore and the Burt filters, and a letter to the manufacturers of these filters will undoubtedly bring a list of the silver-ores now being successfully treated in this way.

The statement that crushing silver-ores in solution is impossible is equally erroneous. Most modern mills crush the ore in solution, in some cases containing as much as 0.12 per cent. of KCN. I cannot for the moment recall any modern mill in Mexico which does not follow this practice. There may be some reason for it in particular instances, but, generally speaking, the practice is to crush in solution. Care in installation avoids any mechanical loss of cyanide, and the chemical loss is so small as to be negligible.

* Received Feb. 2, 1910.

The method of treating sands as indicated by Mr. Bordeaux—namely, first a weak wash, then strong washes, and finally another weak wash—seems to me to be best adapted to particular instances in which it is necessary to avoid large chemical-consumption. The general practice is to use the strong solution immediately after draining the tanks of their filling solution, and to follow with the weak washes.

The statement that the new Pachuca tank treats sand and slime together seems to me to require some modification, since it is only true when the sand has been so finely ground that it may be fairly easily kept in suspension by the agitating-power of the compressed air. So far as I am aware, no attempt has been made to agitate coarse sand. The cost of power and treatment would preclude economical results.

The costs given for crushing and sand-treatment, $4.50 U. S. currency, seem abnormally high. Referring to Ferdinand McCann's book,[1] in which costs are given for many large mills in Mexico, this cost seems to be about half the amount stated by Mr. Bordeaux, and in some cases even less.

At Guanajuato, the use of stationary air-pipes in the bottom of the tank for agitating slime-charges has been discontinued for more than a year.

The reference to the new tanks at Pachuca as "Grothe" tanks is apparently an error, unless some new development has taken place. The general description answers to that of the Brown or Pachuca tank, which was invented and used first in New Zealand, and first introduced in Mexico at Pachuca. The patentee of the tank is represented in Mexico by the firm of Grothe & Carter, of which firm Mr. Grothe is the head. Probably the name Mr. Bordeaux uses was derived in this way. Mr. Bordeaux neglects to state by what means the clear solution is decanted from the Pachuca tank.

In comparing the advantages of shipping bars or precipitates, Mr. Bordeaux has received the impression that the government tax is higher on bars, thus making it advantageous to ship precipitates. This is an error, as I am reliably informed that the government tax is levied on the metal-content of any product, and is thus the same on gold and silver whether in the shape of bars or contained in precipitates, concentrates, or ores. This apparently gives the advantage to the bars, since less weight has to be transported.

INDEX.

[NOTE.—In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics. References to papers expressly treating of the subject named are likewise in italics; and casual notices, giving but little information, are usually indicated by bracketed page-numbers. The titles of papers presented, but not printed in this volume, are followed by bracketed page-numbers only.]

ERRATA.

Transactions, Vol. XXXIX., p. 387, next to last line.

For "28 oz. 1 dwt. of gold" read "28 oz. of silver and 1 dwt. of gold."

3692